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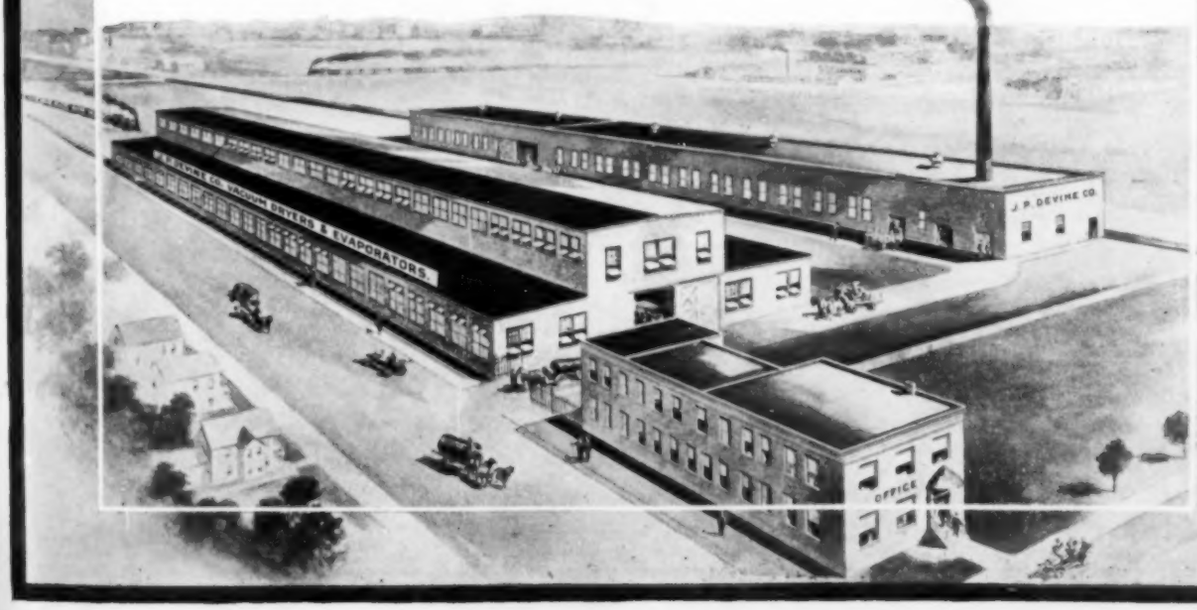

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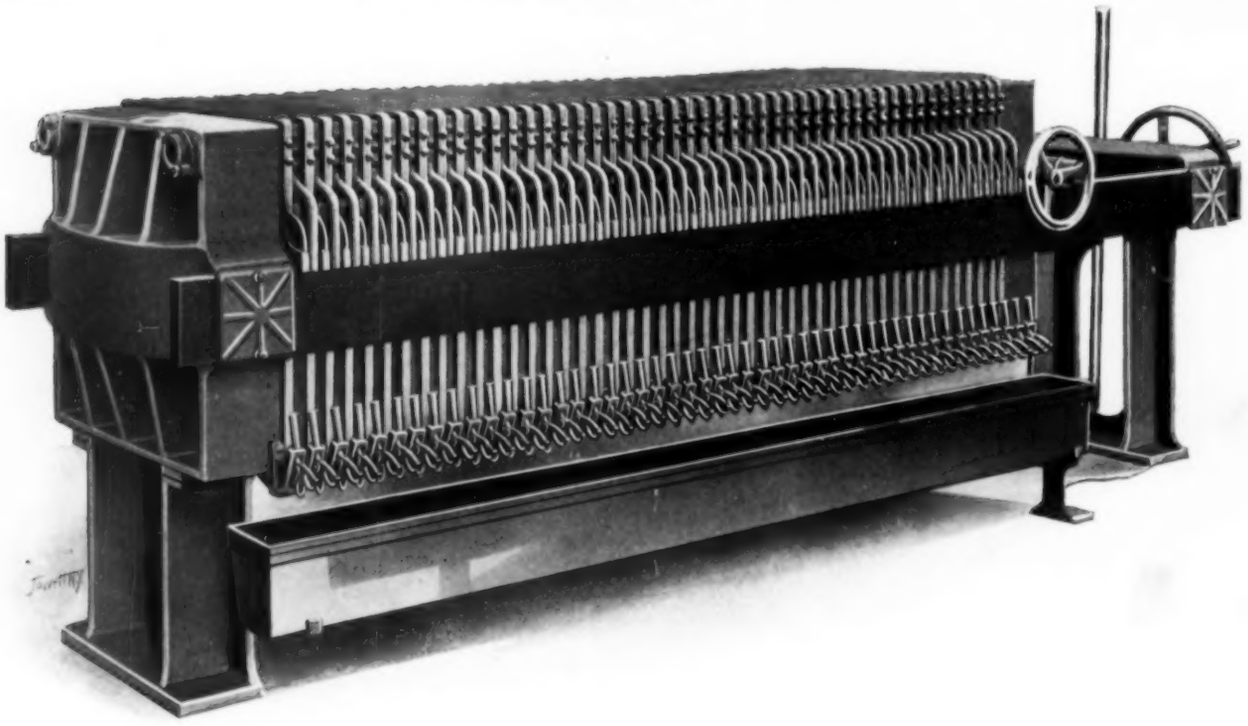

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Steam Distillation. By John E. Teeple 247
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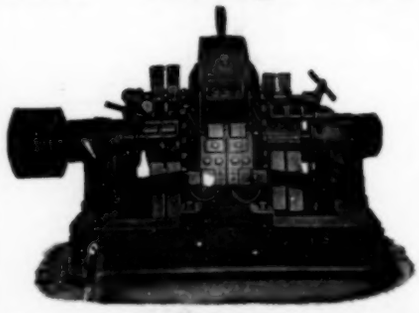

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The Coming Denver Meeting of the American Electrochemical Society

The next meeting of the American Electrochemical Society will be held in September in Denver, Col. During the first eleven years of its life the American Electrochemical Society has become truly international in membership, activity and prestige. The coming years, with the Denver meeting this fall, the proposed meeting in Birmingham, Ala., in the fall of 1914, and a San Francisco meeting in 1915, will give an opportunity to broaden and strengthen the national scope of the society.

In view of such a nation-wide activity the question may be asked: What can electrochemistry do for the country? With respect to the Denver meeting the question becomes more specific: What can electrochemistry do for Western metallurgy? It is natural that the discussion of just this question will mark the keynote of the Denver meeting, a symposium of papers being arranged discussing broadly the potentialities of the applications of electrochemical methods and principles of physical chemistry in Western metallurgy.

If the mountain does not come to the prophet, the prophet has to go to the mountain, and no particular power of prophecy is needed to say that the meeting will be interesting. Electrochemists are born good mixers. Electrochemistry is a borderland of sciences and a borderland of industries. In our age of specialization the scientist and the engineer each uses his own jargon, and so does the physicist, the chemist, the metallurgist and all the other ists of man-made professional distinction, and in this one respect at least modern methods of progress often resemble now the building of the tower of Babel that those who work on the promotion of civilization, each in his own way, no longer understand each other's language.

To remove such man-made gulfs or at least to bridge them firmly, this has been one and perhaps the most glorious function of the American Electrochemical Society. The society has brought together engineers and scientists, to look at each other no longer with distrust, but with mutual respect. Those of its meetings which were devoted especially to particular human mixing reactions, like the Pittsburgh meeting and the recent Atlantic City meeting, have been eminently successful. The Denver meeting should become another milestone in the career of this society.

Electrochemistry and the Electroplating Industry

The symposium on electrodeposition of metals at the Atlantic City meeting of the American Electrochemical Society will add considerably to that Society's already well-deserved reputation as a meeting ground on which men interested respectively in the practical and theoretical aspects of industrial science come together on a basis of

mutual assistance and respect. Such mutual attitude is unfortunately not too commonly met with in the field of applied chemistry, possibly owing in part to the patronizing attitude which too many well-meaning official or academic chemists adopt towards those engaged in industries of which electroplating is typical, in which full advantage is decidedly *not* taken of current scientific knowledge.

The imaginative common sense of the retiring President of the American Electrochemical Society, Professor W. Lash Miller, led him to try the experiment of bringing electrochemists and electroplaters into personal contact and mixing them vigorously in the hope that a solution—or at least an emulsion—might result. If the immediate outcome of the experiment was to demonstrate the surprisingly high values of the surface tensions involved, this merely indicated the desirability of continuing experiments with the view of reducing these. The appointment of a committee to maintain coöperation between the Electrochemical Society and the Electroplaters' Association was the outcome with (approrpriately enough) that master of emulsification, Professor Wilder D. Bancroft, at its head.

This seems to us a decided step in the right direction. It is often too readily assumed, when the application of current scientific knowledge is absent from such an industry, that this is because the industrial men do not want it, and that such application is an easy matter. Both these ideas received a violent refutation at the Atlantic City symposium. The interest and enthusiasm of the attending electroplaters, and their obvious desire to absorb anything electrochemistry could give them were remarkable; if they don't get much benefit from electrochemical science it isn't at any rate because they don't want to.

Even more notable was the obvious gulf between the two parties to the discussion. Not only did they not, as both admitted, "speak the same language"; it was hardly possible for them to come into contact on any point. This gulf cannot be airily disposed of by telling the electroplaters they are on the wrong side; it has to be bridged, and to this end the best joint efforts of both parties on the committee may well be directed. The strongest promise of success in this aim is offered by the spirit of coöperation in which the two organizations have approached the matter.

The first step in the direction of improvement is, however, one which those electroplaters who have not yet taken it can best learn from those who have, viz., the rigid application of method and the systematization of all procedures. We are acquainted with electroplating plants in which methodical supervision and close record-keeping have led by themselves to extraordinary improvements in working. It need hardly be pointed out that this is nothing more or less than the application of scientific method, and that without it an electroplater will look in vain for assistance from scientific knowledge.

Moving Pictures for Industrial Purposes

The photograph has long been recognized as a valuable adjunct to oral and written description. In many cases the simple projection of a lantern slide, or the printed reproduction of a photograph, conveys quickly and definitely an idea which might require a long, involved and less satisfactory verbal explanation. The general adoption of this method in educational and commercial work proves its value. More recently the development of the moving picture has called attention to possibilities in this line which have only begun to be realized. In the field of metallurgy there is an opportunity that should appeal to both educational and commercial institutions, to use moving pictures for more clearly presenting and explaining the operation of many machines which work continuously or in a cycle of intermittent steps.

It is customary in engineering schools for the instructors to illustrate some lectures by means of stereopticon views. The method is excellent, but might be vastly improved by the use of moving pictures. Consider the number of metallurgical operations that could well be illustrated by the latter method: Crushing and grinding, classification of sand and slime, sizing by various forms of screens and the screenless sizer, sinter roasting, concentrating, vacuum filtration by the movable leaf or revolving drum types of filters, handling and conveying materials, charging and discharging blast and retort furnaces by hand or machinery, bessemer converting, dressing amalgamating plates, and so on. The operation of an entire metallurgical plant could be presented on one film, taking the different steps in the process in proper sequence, giving in effect a trip of inspection through the plant.

For the manufacturer of metallurgical machinery, the moving picture may be an adjunct in advertising, promoting and selling. Recently one manufacturer resorted to this method with good effect. His machine was new and little known or understood; it embodied some principles that were novel, and its operation was not easily described; catalogs were inefficient in conveying an idea of its full value. A few hundred feet of film, however, showing the machine in operation, demonstrated quickly and convincingly the nature and scope of the machine and its method of operation. In such cases, simple photographs fail to convey the idea of operation, and leave too much to the imagination. The moving picture, on the other hand, gives the effect of an actual demonstration, and is far more effective than photographs or verbal description. In view of the widespread use of machines for projecting moving pictures, films can be shipped to any important point for demonstration.

Cleaning Filter Cloths.

The prevalent method of treating filter fabrics for the removal of incrustations and deposits of lime and other earthy salts, is with a solution of hydrochloric acid. The method has its disadvantages and is not wholly in favor; filter cloths suffer some deterioration under the action of the acid, and additional tanks may be required. In our issue for last month we noted a patented process for cleaning filter fabrics and screens by the use of a sand-blast. By this method the outer surface of the dried fabric is subjected to the action of the abradant. If the size and hardness of the material used in the blast is carefully selected, and the pressure of the blast properly regulated, it is claimed that the fabric suffers less deterioration than when treated with acid, and that the removal of the incrustation is very effective.

The use of the sand-blast for this purpose is not entirely new, especially in other countries. We believe the idea was first applied in Kalgoorlie, Western Australia, in connection with filter pressing. A great deal of trouble was experienced there with calacerous deposits from the water, which was heavily loaded with salts. Incrustations formed on the plate-and-frame presses to such an extent that after breaking a press it was almost impossible to make tight joints between the members when setting the press up for another run. The deposit resisted removal by scraping or other means, and the sand-blast was called into use with good effect.

From Western Australia the practice spread into New Zealand, where it was used on filter fabrics with more or less success and satisfaction. From New Zealand the method was taken to South Africa. In the latter country the practice is in vogue at one or two mills with fair success. The sand is fed into an open conical hopper which forms a reservoir. A pipe for compressed air extends downward to a point near the apex of the cone, and from the latter point a flexible hose and nozzle conduct the blast to the filter fabric. In the United States the process is in use at the Golden Cycle mill.

Still another proposal for cleaning filters is the use of steam under about ten pounds pressure, assisted by a subsequent wash of hot water. This method has been very effective at one of the cyanide mills in the Black Hills of South Dakota, having been introduced there by Mr. F. C. Bowman. The filter is of the drum type. When the cloth needs cleaning, steam is forced through successive sections of the filter by means of the pipe ordinarily used to conduct compressed air to dislodge the cake. The effect of the steam is to immediately loosen and disintegrate the deposit of lime, after which it can easily be washed out by hot water, leaving the meshes of the cloth clean and free.

Rolled Steel and Cast Iron

It is interesting to note that common knowledge and such statistics as can be cited unite in testimony that the iron casting is decadent as compared with its vigorous rival in the ferrous material field, rolled steel. Continually one hears of new uses to which steel is put, and practically all these uses involve rolled steel. Metal furniture employs rolled steel, the automobile carries much rolled steel, a little forged steel and an occasional steel casting, but a negligible amount of cast iron. Sheet piling, concrete reinforcing bars, relatively new products, are of rolled steel. The cast-iron column is decadent in building construction, while the skeleton structure of rolled steel thrives amazingly. The cast-iron carwheel is losing ground to steel, though in this case the steel product is either a real forging or is rolled in a manner which does not place it within the category of the ordinary rolled-steel product. In the majority of forms of hardware rolled steel has long been replacing cast iron.

A few references like these are amply suggestive of the rapid march of rolled steel, and of the relative, if not absolute, decadence of the iron casting. Inasmuch as the total production of ferrous material, measured by the production of pig iron, which is a fair index despite the use of a moderate amount of old material, has been increasing rapidly, doubling in a general way every decade, it requires close scrutiny to determine whether the decadence of the iron casting is absolute or merely relative to rolled steel. There have never been statistics of the production of iron castings, but the production of foundry iron should furnish a valuable clue, for the consumption of cast-iron scrap in the foundry cannot have increased very rapidly. The full statistics of pig-iron production by grades in 1912 are not yet available, but there are statistics which segregate Bessemer, basic and manganese bearing metals from other pig iron, and these show that the production of foundry, malleable, forge and miscellaneous grades was slightly less than 6,500,000 gross tons in 1912, whereas in both 1910 and 1909, the next best years in total pig-iron production, the output of these grades was about 6,750,000 tons. The total production, however, has been: 1909—25,795,471 tons; 1910—27,303,567 tons; 1912—29,727,137 tons. Here was a large increase in total pig iron, and a decrease in the foundry grades.

The statistics of the market seem to bear the same testimony. In the general revival in the iron and steel industry in 1912, rolled steel products advanced until they reached a certain level, early in October. Since then the level has been maintained, there being practically negligible advances and declines. Foundry pig iron, on the other hand, after reaching a high point in December, has steadily declined in price, and in the past month the declines have been particularly rapid. With quoted prices unchanged, the steel mills have month by month been realizing better prices on their shipments, as old and lower-priced contracts were worked out, while at the same time pig iron, which has certainly been subject to no adverse influence from the steel trade, has been declining.

It is one of the adages in the iron and steel trade that in case of a general improvement in the industry the foundry trade finds its improvement about six months later than the steel trade, lagging behind it by about this distance. Before this time, therefore, there should have been a very distinct improvement in the foundry trade, which one would naturally expect to find in its reflection in the market prices for pig iron.

It is sometimes suggested that the iron casting is yielding to the steel casting. As a generalization, this is altogether a misconception, though it is undoubtedly true in a few specific instances. One must compare totals to reach a proper conception. As long ago as in 1906 the production of steel castings amounted to 773,705 gross tons, while in that same year the production of forge pig (of which but a very small tonnage really reaches the puddling furnace), foundry iron and malleable iron, exceed 6,000,000 tons, and making allowance for the consumption of scrap the production of iron castings undoubtedly much exceeded this figure. Yet it is a fact that, with 1912 still to be reported upon, there has been no year in which the production of steel castings has reached a million tons, showing that the tonnage gain since 1906 has been altogether insignificant. Thus the steel casting has not replaced the iron casting to any appreciable extent. The statistics for 1912 when presented will undoubtedly show that in point of percentage the steel ingot has grown much more since 1906 than the steel casting.

Readers' Views and Comments

"Thermistry" and the "Thermist"

To the Editor of Metallurgical and Chemical Engineering:

Sir:—What shall we call the man who has charge of the heat treatment of steel or for that matter of any metal or alloy? He is hardly a metallurgist or metallurgical engineer although those terms are sometimes applied. Neither is he a chemist, although the chemist often has charge of heat treatment; yet the man in charge of the latter may know very little chemistry.

Here then is a distinct position or grade which is becoming more and more important as we more fully understand the possibilities of this branch of metallography. Then why not call it by a distinct name?

Thermistry then, is the art and science of heat treatment and heat measurement and the practitioner would be a thermist. The name is descriptive, dignified, concise, and much less awkward than "metallurgist in charge of heat treatment" or "engineer in charge of heat treatment."

H. M. Boylston.

Cambridge, Mass.

Volatilization Processes in the Metallurgy of Gold

To the Editor of Metallurgical and Chemical Engineering:

Sir:—On page 220 of the April, 1913, issue of Metallurgical and Chemical Engineering, there appears a review of an article originally published in the monthly *Journal* of the Chamber of Mines of Western Australia regarding the recovery of gold by volatilization. In the original article the author, Mr. Ben Howe, states that his process merely takes advantage of a fact long known to metallurgists of the old chlorination school, viz., that when gold ores are roasted with salt the precious metal is volatilized in greater or less degree, according to the conditions of temperature, time and composition of the roasting mixture. By determining the proper conditions for the roast, Mr. Howe finds that he can volatilize the gold to such an extent as to constitute an efficient method of extraction. He further observes that this is "a process, the application of which, at any rate, is new to the metallurgy of gold."

It has been remarked that methods and machines that are old and well known to one generation or to one part of the world are rediscovered and revived by later investigators or in another section. The present instance is a case in point. About 1897 or 1898 Mr. Edwin C. Pohlé and Mr. Stuart Croasdale, then respectively chief assayer and chief chemist for the Globe Smelting & Refining Company, Denver, Col., devised and patented a process based on the same principle as that suggested by Mr. Howe. Mr. Pohlé was first led to investigate the volatilization of metals as a result of serious losses encountered in assaying high-grade silver ores from Aspen, Col. When an assay was made with the customary "salt cover" wide variations occurred in the results, which, however, were more uniform when no salt was used. The two gentlemen named then began a serious investigation of the subject, with the idea of using volatilization in the recovery of metals, particularly from refractory ores.

They found that gold, silver, lead, and copper could be almost completely volatilized from most ores. It was necessary to have in the ore some form of sulphide, in proper proportion to react with the added sodium chloride. The reaction liberated nascent chlorine which combined with the metals to form volatile chlorides. If the ore contained a high percentage of sulphide it was first partly roasted; if it contained none, or not enough, raw pyrite was added. The laboratory tests were simple so far as volatilization was concerned, but recovery was a different matter. The process was finally worked in an experimental plant in Denver, and later a commercial plant was erected at Mayer, Ariz. The latter was never operated, as far as I am aware.

It may be surmised that the stumbling block to success was the recovery of the volatilized chlorides. As one metallurgist remarked, it was difficult to see the wisdom of converting the metals into a less tangible condition than that in which they already existed in order to attempt their recovery. Bags, sprays, solution towers, etc., were tried at the recovery end, but without success. The problem was never satisfactorily solved. The volatilization feature, however, was attractive in regard to refractory ores, for they usually yielded to the treatment.

There is one possible solution of the recovery problem which has occurred to me since the development of the Cottrell method of fume condensation by means of a high-potential electric discharge. The volatilized metal chlorides might be suitably condensed by the Cottrell process, forming a concentrate of mixed chlorides, which could then be separated by chemical means. In the absence of any information as to how the Australian metallurgist is to recover his chloride fumes, it will be interesting to await further developments. His process, however, certainly is not "new to the metallurgy of gold."

Denver, Col. H. C. Parmelee.

Simple Rules for Temperature Conversion

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I am interested to read Dr. Hering's communication in your April issue (p. 176), on mental calculations from the Centigrade to Fahrenheit thermometer scales and vice versa. He omits, I think, one for which I claim no originality, but find convenient. It is not approximate, but exact.

In either case, whether Centigrade or Fahrenheit, add 40 to the reading, then if Fahrenheit take five-ninths and subtract 40, or if Centigrade take nine-fifths of the sum and subtract 40.

My attention was called to this by one of my students, and I find it a simple method.

D. H. Childs.

Technical High School,
Buffalo, N. Y.

* * *

To the Editor Metallurgical and Chemical Engineering:

Sir:—As many of your readers know, the conversion of daily temperatures in Centigrade degrees into Fahrenheit degrees is the bug-bear of Americans traveling in Europe, and who has not seen the letter published daily in the Paris edition of the New York *Herald* about to the following effect?

"Dear Editor:

Can you furnish me with a simple rule for converting centigrade temperature into Fahrenheit?

"Most sincerely yours,

"Young Philadelphia Lady."

Your last number contained a gallant attempt at a solution of this perennial problem, signed oddly enough "Philadelphia," but alas it was not intended for use in these latitudes, and we hope and trust that the "Young Philadelphia Lady" may never experience, here or hereafter, the temperatures for which Dr. Hering gave his approximate but very convenient rules. No; for purposes of discussing the weather, the problem has not yet been solved, except by the time-honored expedient of memorizing a few fixed points, such, for example, as that 10° Centigrade is 50° Fahrenheit and every 10° Centigade more or less is 18° Fahrenheit higher or lower than 50° Fahrenheit, every 5° Centigrade is 9° Fahrenheit, and every 1° Centigrade is 1.8° Fahrenheit. This simple mental exercise solves all the weather problems.

South Bethlehem, Pa. Jos. W. Richards.

Alumina from Clay by the Fluoride Process

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have been much interested in the symposium of papers on alumina in your March issue and would like to add something to the discussion.

When a mixture of aluminium fluoride and clay, in such proportions that atoms of silicon and fluorine are to each other as one to four, is heated to a red heat in the presence of water vapor, silicon fluoride passes off and alumina is left.

The only surprising thing about this is the completeness with which this occurs, and the small excess of fluorine needed to secure the complete elimination of silica. In a previous paper (Eighth International Congress of Applied Chemistry, Inorganic Section, Original Communications, Vol. II, p. 67, and Vol. XXV, p. 115), under the title "Action of Some Fluorine Compounds on Clay, Water, etc." I gave results of tests made on some of the details of the process of producing alumina from clay in the above manner. The resulting silicon fluoride reacts with steam or a water spray, the silica is separated by centrifugal action, and the gaseous hydrofluoric acid passes up through a tower filled with clay, over which a small stream of water is flowing. The acid solution from the tower is treated with excess of mildly calcined clay, precipitating the excess silica, much of the iron and titanium, and regenerating the aluminium fluoride.

Two important questions—after the one, "Will it work?"—are, first, the cost of the alumina, and second, the purity. And in that order, because two-thirds of the alumina comes from the calcination of the aluminium fluoride by the reaction

$$Al_2F_6 + 3H_2O = Al_2O_3 + 6HF$$

and that fluoride of aluminium can be made of good purity by proper neutralization with clay.

While no claim is made that metal for electrical conductors can be made by this process because small amounts of impurities so greatly affect the conductivity, yet the other uses are few for which such metal cannot be properly recommended.

Further on the subject of purity, I find that half the titanium and more than half the iron of the mixture are volatilized with the silica in the calciner. As to costs the first point must be the recovery of the fluorine. From 80 per cent to 90 per cent has been recovered in the laboratory, and probably that figure will stand until some larger tests are made.

On the 80 per cent basis, calculations show a cost per ton, including the factors of labor, power, fuel, acid and fluorspar in the neighborhood of $17. Appreciating the doubt that properly clouds all calculations based on laboratory tests, it is nevertheless interesting when those calculations give a low value.

We are now building a calcining and absorbing set to try this out on a larger scale.

Attempts have been made to recover the needed fluoride from the gases which are given off in the manufacture of phosphate fertilizer.

These attempts have been frustrated so far by lack of settling space for the dust arising when the ground phosphate is dumped into the bins with the acid. Very little work has been done on this, however.

D. H. CHILDS.

Technical High School,
Buffalo, N. Y.

Segregation and Deoxidation and the Efficiency of Aluminium, Silicon and Titanium as Deoxidizers.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—To supplement the notes on the discussion of sound ingots before the American Institute of Mining Engineers, as reported in your March issue, p. 151, the following remarks which I made in the discussion of Dr. Dudley, seem pertinent.

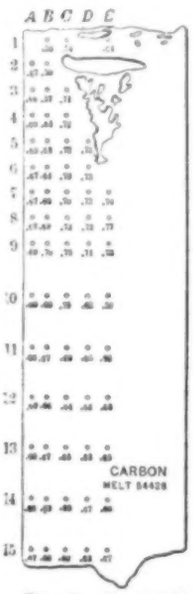


FIG. 3.—CARBON-DIAGRAM OF INGOT AND BLOOM.

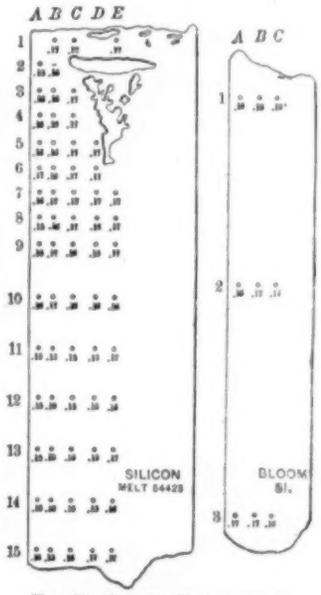
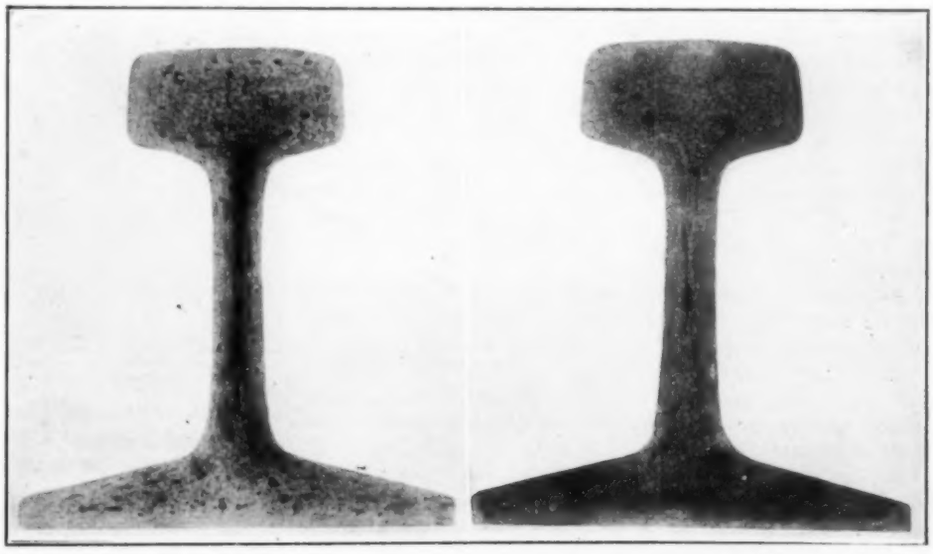


FIG. 7.—SILICON-DIAGRAM OF INGOT AND BLOOM.

FIGS. 3 AND 7 OF DUDLEY PAPER



SULPHUR PRINT STANDARD OPEN-HEARTH

SULPHUR PRINT TITANIUM OPEN-HEARTH

In reading the very interesting paper of Dr. P. H. Dudley I noticed on the carbon diagram of an ingot the following carbon determinations:

In Fig. 3, line 9, a normal segregation of from 69 to 73; line 10, a negative segregation of from 69 to 59; line 12, a negative segregation of from 69 to 63; line 13, a negative segregation of from 68 to 65; line 14, a negative segregation of from 68 to 60.

I will endeavor to explain why this negative segregation occurs in the lower half of the ingot and submit as a partial explanation the fact that the segregation is induced by the oxides in solution upon which the carbon of the steel reacts.

In examining the silicon diagram of the same ingot as shown in Fig. 7, it will be found that the silicon is nearly uniform. If we admit for a moment that there are some oxides in solution in the center of the ingot and I will try to show later that it is proper that we should, why were these not reduced by the remaining silicon which is supposed to be such an efficient deoxidizer?

This brings us to a discussion of the comparative efficiency of the three deoxidizers mentioned by Dr. P. H. Dudley and Mr. Talbot, viz.: aluminium, silicon, and titanium.

Aluminium as a Deoxidizer.—Aluminium is the most powerful of all known deoxidizers. It will reduce the oxides of both silicon and titanium, but unfortunately by oxidation it will be converted into alumina, which melts at 2010 deg. C. and consequently is infusible at the temperature of the bath. This fact has received only slight consideration in many quarters.

This infusible alumina will remain in the steel and if some part of it by chance is near the top of the ingot it will cover the surface of the pipe and shrinkage cavity, eliminating any advantage which could be gained from immediate rolling, as suggested by Mr. Talbot. It will be noticed that Mr. Talbot has suggested in "well made steel" an addition of 2 oz. of aluminium per ton, or in the case of a 12,000-lb. ingot an addition of 12 oz., which by oxidation will produce 24 oz. of alumina per ingot; this most infusible material will be disseminated throughout the mass in small shot, which later on will be rolled into streaks.

When such streaks of alumina are present between the metallic crystals of a steel rail they destroy the continuity of the metal, having no strength in themselves they are very readily broken and cracks started from this cause are bound to spread through the adjacent metal when it is subjected to the severe stresses and strains of the heavy wheel loads passing over the rail.

Silicon as a Deoxidizer.—Silicon, as already mentioned, is not a powerful deoxidizer and when added to steel it leaves a remainder in it. Usual determinations of silicon give no clue as to the state in which it is found in the metal, whether as silicides, silicic acids or silicates, but if it exists in either of the two latter forms its presence is exceedingly harmful.

Microscopic examination of rail steel discloses small globules of foreign matter consisting of sulphide of manganese and also of oxides and slags. It has been noticed that very often a steel containing as much as 0.10 per cent of silicon in the form of iron silicide will be wild when poured necessitating an addition of aluminium as a deoxidizer in the molds.

Titanium as a Deoxidizer.—Titanium added to the steel in the ladle, after the silicon, will complete the deoxidation. Heats with an addition of titanium of from 0.05 to 0.10 per cent will not require any additional deoxidizer in the molds.

It is the only deoxidizer known which can be used without danger of leaving any of the products of its oxidation in the bath of steel, as pointed out in Dr. P. H. Dudley's paper.

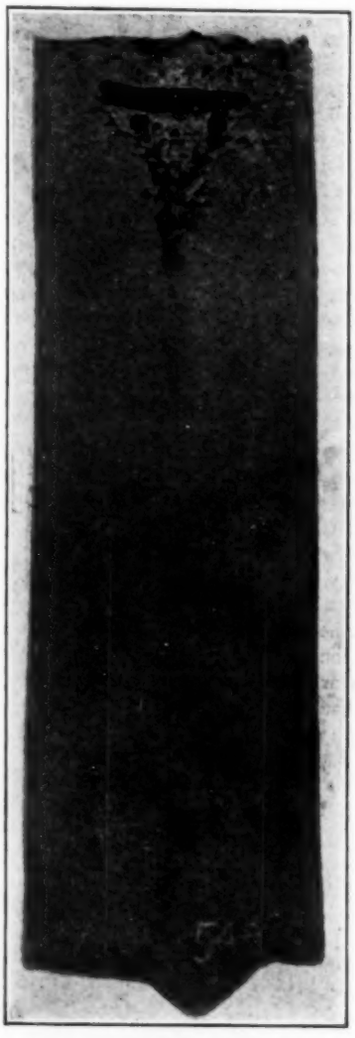
By oxidation titanium produces titantic acid which will flux the slag in suspension in the steel and if sufficient time be given for the reaction to be completed in the ladle, after its addition, all of the products of its oxidation will rise to the surface leaving a clean steel.

If in the Talbot process such a deoxidizer be used to make pipeless ingots, the pipe being clean will weld more readily when the ingot is compressed.

If a comparison were made between two sulphur prints from the cross section of two rails coming from the same part of two different ingots; the rails being of substantially the same chemical composition but the one having been properly treated with 0.10 per cent of metallic titanium, the other being a plain open-hearth steel, the former would not show any slag inclusion and only a very slight segregation, while the latter would in many instances show a dirty steel "full of slag" and an excessive amount of segregation. This is evident from the adjoining comparative sulphur prints.

It will be noticed that the untreated rail has a silicon content of 0.15 per cent and it is a fact that a few pieces of aluminium were used in the mold in the production of this steel. If, therefore, silicon is a powerful deoxidizer why should there be such an amount of oxides and slags and consequently so much segregation in this steel? The sulphur print of the rail treated with ferro-carbon-titanium does not show anything of the kind and it is therefore reasonable to assume that the elimination of the oxides will result in diminished segregation.

From the foregoing my conclusion is that for the manufacture of good rail steel silicon, a cheap deoxidizer, ought to be used to a limited extent, but that deoxidation should be completed with titanium, which as mentioned by Dr. Dudley will not only react on the oxides, fluxing the slag, etc., but will also reduce the iron silicates formed in the steel by the previous introduction of ferrosilicon.



SPLIT INGOT

As to the impression that rail steel treated with ferro-carbon-titanium pipes, I will say that good steel always pipes and in experimenting along this line I have found, although I cannot as yet give a positive explanation for the fact, that silicon and titanium are antagonistic and that the silicon content in rail steel treated with titanium should be reduced within the limits of from 0.06 to 0.10 per cent, for if the silicon is so reduced it will be found that the shrinkage cavity will be similar to the one shown in Dr. Dudley's paper (see adjoining illustration) and that the cavity will be clean and free from all silicates, slag or alumina so as to increase its welding properties when the ingots are immediately bloomed as suggested in Dudley's and Talbot's processes.

The drop and exhausted ductility test, for basic open-hearth steel rails is excellent and we are indebted to Dr. P. H. Dudley for its introduction. Very often, however, a rail rolled from a heat treated with aluminium, in accordance with the ordinary practice or by the Talbot or Hadfield process, might pass the drop test to the entire satisfaction of both the railroad and the mill, but as already explained when I mentioned aluminium as a deoxodizer, such a rail containing streaks of alumina, after a certain length of time in service will develop defects which are very often the real origin of broken rails and consequent wrecks.

I wish to state, however, that I do entirely approve of Dr. P. H. Dudley's specification for the manufacture of basic open-hearth rails but will add, as already explained, that if no aluminium be used (which transformed into alumina will coat both the shrinkage cavity and the pipe) the steel bloomed under its equalized initial heat will be rendered pipeless by the usual discard of from 8 to 10 per cent.

Niagara Falls, N. Y. N. Petinot.

Nichols-Hesse Dinner

On the evening of Saturday, April 19, a complimentary dinner was given by the chemists of the United States to Dr. William H. Nichols, the president of last year's International Congress of Applied Chemistry, and to Dr. Bernhard C. Hesse, the Secretary of the Congress, as a mark of their appreciation and affection. The dinner was held at the Chemists' Club in New York City, and as some 225 men had come to do honor to the two men who more than any one else bore the brunt of the battle and made the Congress a success, the large lecture room (Rumford Hall) had been chosen for the occasion. It was not by any means merely a New York City affair. The names of the diners as well as of the speakers showed that American chemistry was broadly represented.

That the Chemists' Club as a social institution is fully up to date in every respect, was evidenced by the fact that just before the beginning of the dinner the extra waiters which had been engaged for the evening went on a strike. But it is also a fact that the House Committee settled the strike quickly and that very few of those who sat down to dinner in the prettily decorated hall realized what a narrow escape there had been from a somewhat awkward situation. Well, everything went along lovely. The dinner was excellent and greatly enjoyed. It was simply representative of the potentialities of the cuisine and the wine cellars of the Chemists' Club on one side and of the congeniality of American chemists on the other.

WILLIAM H. NICHOLS

BERNHARD C. HESSE

After the dinner was over, the assemblage was graced by the presence of a number of ladies gathering on the balcony of the banquet hall. The speechmaking began and lasted till after midnight.

Dr. **Chas. Baskerville,** of the College of the City of New York, for the Committee of Arrangements, introduced Dr. **Edward W. Morley,** of West Hartford, Conn., as the presiding officer of the evening. Dr. Morley who had been the Honorary President of the Eighth International Congress, made a felicitous little speech and concluded by introducing in turn Mr. **Arthur D. Little,** of Boston, Mass., as the toastmaster of the evening.

Mr. Little spoke humorously and feelingly of the achievements of the Congress and of the rôle which the personal element had played in it, of the work of Nichols and Hesse, and of the tragic death of Morris Loeb.

At the conclusion of his introductory speech Mr. Little read a long series of messages, including letters from Dr. Chandler, Dr. Remington, Mr. Kuttroff, Mr. Adamson, Dr. Bogert, Dr. Cameron, and Mr. Wm. J. Matheson. Mr. Matheson had been the treasurer of the Congress and a toast was drunk to him on his success. As Mr. Little explained this was one of the rare occasions of this kind which ended with a surplus instead of a deficit. This result was mainly due to Mr. Matheson's highly effective work as treasurer. At the concluding meeting of the organization of the Congress, held just the night before, $2,000 out of the surplus had been granted to the Chemists' Club.

Mr. **C. O. Mailloux** read some twelve official letters of congratulations to Drs. Nichols and Hesse from French and Italian societies.

Then the speech making proper began. Dr. **Wilder D. Bancroft,** of Cornell University, spoke as Past President of the American Chemical Society and of the American Electrochemical Society. He spoke exceedingly well, both concisely and to the point. Starting with a funny church collection story, he passed to a review of the success of the Congress—for it was a great success—and of the men who made it such. Morris Loeb lost his life at it. Dr. Rosengarten nearly did. Mr. Matheson produced a surplus and Dr. Hesse gave up a year and a half of his life to produce thirty volumes of Transactions and a great many tons of literature. And as to Dr. Nichols' work as president, if we should ever go again through such a Congress in our life-time (which Heaven forbid) and if we should be forced to get along without Dr. Nichols as president (which Heaven forbid)—we might think in poetry:

Twinkle, twinkle little star—
Riding on a trolley car—
Trolley car went off the track—
Gee, I wish we had our Nickels back.

The next speaker was Dr. **William Brady,** of the Illinois Steel Company, of South Chicago, Ill., who represented the American manufacturers. He spoke of the American manufacturer as a business man. His first concern is the quality of his product and he educates the American public in quality. Then he looks for quantity and systematizes and organizes. Now he is very much interested in his men and their safety. If it is possible to go to extremes, the American manufacturer is willing to go to extremes in the matter of safety of his employees. Dr. Brady concluded with a happy reminiscence from early life in a country school.

Mr. Little then introduced Dr. **Leo. H. Baekeland,** of Yonkers, N. Y., as Past-President of the American Electrochemical Society and of the American Institute of Chemical Engineers (" the only aristocracy among American chemical bodies"), as the discoverer of velox paper and the "anonymous discoverer of bakelite," "but the discovery which has most endeared him to his friends was his discovery of Mrs. Baekeland."

Dr. Baekeland in his happy speech asked: "Was it worth while?" Answers to questions as to worthwhileness depend on the point of view. If different viewpoints were not possible, if viewpoints could not be exaggerated, honest men could not differ in discussions at all. But as applied to the past Congress, different viewpoints were hardly possible. He pitied anybody for whom the Congress had not been very much worth while. Of course, those will be disappointed who want to get more out of the world than they put into it. The man behind the machinery and organization was Dr. Nichols. Dr. Nichols' work at the Congress has been a school for every one of us. "I had a bully good time."

Dr. **David T. Day,** of Washington, D. C., spoke of the Congress as a lightning express on its way to Russia. Hesse was at the switch and got the proceedings of the Congress out before the Congress began to proceed. It was a wonder what it is possible to get out of a Hessian crucible. The calm spot in the center of the cyclone was Dr. Nichols. The inspiration came from him. He put the different men together and made them work, and while he worked very hard in doing so, he was not used up himself. That is, he was the catalytic agent. Nickel as a catalytic agent may tire at times. Nichols does not, being a plural and representing a polymerization product, Dr. and Mrs. Nichols. In order that Dr. Nichols and Dr. Heese should take with them into later life the memory of the appreciation and affection of their American fellow chemists Dr. Day presented, in the names of the American chemists, to Dr. Nichols a counterpart in silver of the desk set in the Royal Treasury in London and to Dr. Hesse a silver loving cup.

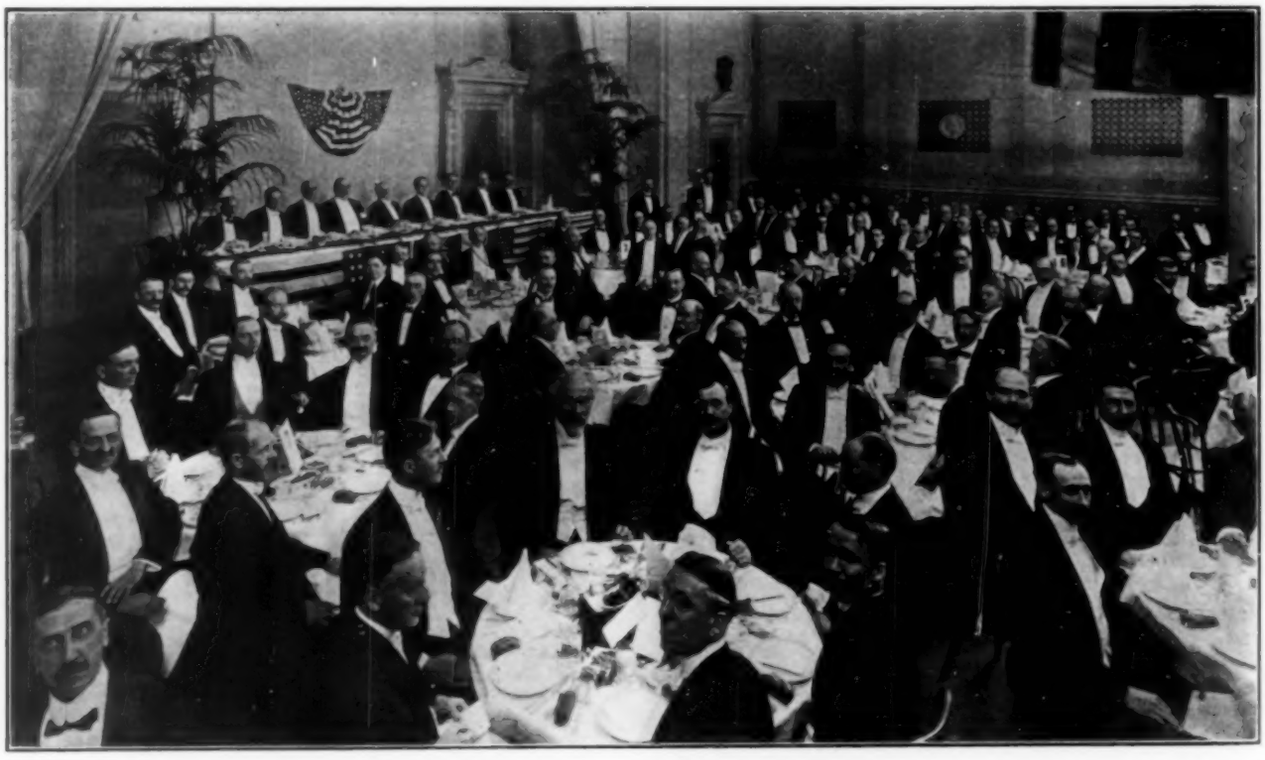
Dr. **William H. Nichols,** in his gracious words of appreciation, pointed out as one great lesson of the Congress the enormous amount of ability and chemical knowledge stored up in our universities, colleges, laboratories all over the country, and the enormous amount of hard and untiring work done there in a quiet and unostentatious way. He pointed out that this work needs more encouragement. He pledged his aid to render it possible in future to make the most of this work of college professors for the good of their students and the good of the world. This being a true problem of "conservation," nothing of these vast intellectual resources of the country should go to waste.

Dr. **B. C. Hesse** had before him a complete bound set of the *Transactions* and told something of the troubles which he had to meet as the secretary of the Congress. But from his statistical figures it seemed that out of eight there were seven men satisfied and only the eighth one was disappointed. The documentary evidence produced by Dr. Hesse as to the reasons why some were disappointed was truly amusing. "Get the eighth fellow to do as the rest of us do." Dr. Hesse concluded with hearty words of thanks to Dr. Nichols and to the chemists of the United States.

A toast drunk to the president of the Ninth International Congress of Applied Chemistry, Dr. Walden, in St. Petersburg, concluded the evening, which will long be remembered.

In the following we give the alphabetical list of those who participated in the celebration in honor of Dr. Nichols and Dr. Hesse:

L. Albrecht, C. Amend, O. P. Amend, T. Armstrong, F. W. Atkinson, W. S. Ayres, L. H. Baekeland, F. H. Baldwin, W. D. Bancroft, C. Barkerville, H. P. Bassett, G. C. Bauer, H. V. Berg, W. D. Bigelow, M. W. Bingham, H. B. Bishop, F. Black, V. G. Bloede, E. Boise, S. Bookman, F. C. Bowman, W. Bowman, C. S. Bradley, W. Brady, A. A. Breneman, I. H. Brickenstein, A. von Briesen, T. L. Briggs, C. A. Browne, H. Buel, C. F. Burgess, C. F. Chandler, S. R. Church, C. B. Clark, V. Coblentz, E. J. Cole, A. H. Cowles, H. N. Cox, F. D. Crane, A. S. Cushman, W. H. Davis, D. L. Davoll, D. T. Day, E. deMeeus, C. Dill, R. E. Doolittle, C. A. Doremus, J. Douglas, L. A. Dreyfus, H. DuBois, T. R. Duggan, B. W. Dunn, A. W. Elkins, W. H. Engels, W. H. Erhart, W. J. Evans, H. A. Fales, W. C. Ferguson, C. G. Fink, T. B. Freas, H. H. Fries, M. M. Garver, J. F. Geissler, A. E. Gibbs, L. Gifford, T. T. Gray, W. S. Gray, E. V. Grano, A. Gref, A. P. Hallock, C. Haner, E. Hart, J. Hasslacher, F. R. Hazard, F. Hemingway, E. Hendrick, J. B. F. Herreshoff, C. H. Herty, A. E. Hill, N. Hill, Jr., H. W. Hillyer, K. J. Holliday, F. Holzrichter, W. D. Horne, J. W. Hornsey, H. Howard, A. Humphreys, H. A. Huston, A. von Isakovics, M. Ittner, F. Jaeger, A. B. Jones, C. M. Joyce, J. Kahn, W. C. King, P. C. Kingsbury, E. D. Kingsley, E. C. Klipstein, D. P. Knowland, A. E. Kraemer, O. H. Krause, H. J. Krebs, G. F. Kunz, A. Kuttroff, K. B. Lamb, N. Lane, J. Langeloth, A. C. Langmuir, N. A. Laury, A. D. Little, E. G. Love, F. S. Low, B. A. Ludwig, C. S. Lutkins, K. G. MacKenzie, C. O. Mailloux, J. R. Marble, S. C. Mastick, W. J. Matheson, H. F. Merriam, H. A. Metz, F. J. Metzger, D. Miller, W. L. Miller, H. I. Moody, R. W. Moore, S. R. Morey, J. L. Morgan, L. Morgan, E. W. Morley, R. K. Murphy, I. J. R. Muurling, W. S. Myers, H. S. Neiman, A. C. Neish, C. W. Nichols, F. H. Nichols, W. H. J. Nichols, J. H. Nield, S. J. Norris, G. Ornstein, O. Palmenberg, F. J. Parker, C. L. Parsons, E. D. Pearce, G. H. Pearson, J. C. Penny, C. Pickhardt, W. P. Pickhardt, L. Pitkin, A. Plaut, C. E. G. Porst, M. R. Poucher, F. H. Pough, T. W. Pritchard, J. de Pulligny, A. M. Purves, J. W. Richards, C. E. Rigg, A. E. Rising, A. Robb, E. F. Roeber, A. Rogers, G. D. Rosengarten, C. Roth, W. E. Rowley, H. D. Rahm, L. Saarbach, M. F. Schaak, E. Schill, J. D. Schwentzman, E. W. Seabury, H. C. A. Seebohm, F. Seeler, R. A. Shaw, A. Smith, F. E. Smith, T. E. Smith, E. A. Sperry, J. Speyer, J. W. Starr, S. Steele, G. C. Stone, I. F. Stone, E. H. Strickler, E. Stütz, O. Sussman, C. E.



PHOTOGRAPH TAKEN AT NICHOLS-HESSE DINNER

Thies, B. F. Thomas, G. W. Thompson, M. Toch, C. Travelli, S. A. Tucker, H. C. Tuttle, A. P. Villa, T. B. Wagner, L. I. Waldman, W. H. Walker, J. M. Weiss, E. Weston, M. C. Whitaker, L. R. Whitcomb, C. T. Whittier, F. G. Wiechmann, H. Wigglesworth, H. A. J. Wilkens, G. F. Willett, F. Wyatt, W. D. Yearsley, C. B. Zabriskie.

International Geological Congress

The Twelfth International Geological Congress will meet in Toronto, Canada, from August 7 to 14, 1913. The headquarters will be at the University of Toronto.

The membership fee is $5.00.

Titles of papers and proposals to be presented should be submitted to the General Secretary, R. W. Brock, director of the Geological Survey of Canada, Ottawa, Ont., Can., as early as possible. Contributions must be received by May 1st if they are to be printed in time for distribution before the meeting.

The official language of the Congress is French, but contributions may be submitted in French, English or German.

The following topics have been selected by the executive committee as the principal subjects for discussion: Coal resources of the world; differentiation in igneous magmas; the influence of the depth of the character of metalliferous deposits; the origin and extent of the pre-Cambrian sedimentaries: the sub-divisions, correlation, and terminology of the pre-Cambrian; the extent of which the ice age was broken by inter-glacial periods; the physical and faunal characteristics of the palaeozoic seas, with reference to the value of the recurrence of seas in establishing geological systems.

A great many different excursions have been arranged, preceding the sessions of the Congress, during the sessions, and after the sessions. Details concerning the same may be found in the second circular issued by the Congress. This circular as well as all other information may be obtained from the Secretary of the Twelfth International Geological Congress, at the Victoria Memorial Museum, Ottawa, Ontario, Canada.

Electrochemical Spies in Niagara Falls

A mild sensation has been recently caused in Niagara Falls electrochemical circles by the capture of a Norwegian engineer in the act of making drawings and notes in one of the larger plants. An examination of the man's effects brought to light data from a number of plants in Niagara Falls and elsewhere.

His mode of procedure was to get a job in a repair gang thus securing access to more departments than would be possible in special work. Two accomplices went about the country with this chief, following the same lines.

A confession by a compatriot in one of the plants brought out the fact that money was plentiful and was freely used where necessary. From the same source it was hinted that large foreign interests were responsible for the presence of these spies and financed the operations liberally. One of the names used at Niagara Falls by this man was Kverne, those of the assistants, a Swede and a German, not yet having been obtained.

Publicity is given these facts as a warning to manufacturers, as the considerable ability of this engineer, evidenced in his notes, makes his presence dangerous in plants where secrecy is considered essential.

Closing of Norwegian Electric Iron Ore Reduction Plant

The Electric Iron Smelting Company Hardanger, of Norway, has been forced, after three-quarters of a year's operation, to close down its factory, according to information published in Scandinavian newspapers. The iron smelting process used by the Hardanger company was the well-known electric shaft furnace process developed by the Swedish engineers, Gronvall, Lindblad and Stalhane, and which is in successful operation at different steel plants in Sweden.

The news that the Hardanger plant has been closed may appear startling, contrasted with the successful operation of furnaces of the same type in Sweden. The reason is due to inavailability of suitable raw materials in Norway. The Swedish electric-blast furnace has to be charged with charcoal, which is readily available in Sweden, while the Norwegian plant was using coke, since charcoal was not obtainable at the plant at a reasonable price.

During the experimental work of developing the electric-shaft furnace at Trollhättan coke was found unsatisfactory in the operation of the furnace, so the closing of the Hardanger plant only confirms earlier results.

The Hardanger company owns the water-power which was used for the electric smelter, amounting to 14,000 hp, and have leased it to a French company, which will utilize it for electrochemical industries.

The Non-Ferrous Metal Market

Since our last report the non-ferrous metal market has been affected by industrial and political conditions both in this country and abroad. Floods and the prospect of tariff changes have combined to affect the domestic spelter market; Mexican revolution has resulted in reduced exports of lead from that country to Europe, with a consequent advance in the foreign market, which was later accentuated by labor strikes in Australia; the tin market has been subject to wide fluctuations, but the copper market has become steadier.

Copper.—The depression which was manifest in March was removed in April. The influence of heavy buying in Europe was felt in the domestic market, where most of the business was for foreign trade. The statistics for March, which were published early in April, were rather favorable, and the situation is considered much improved. The last quotations are 15.35@15.55 cents for Lake, and 15.20@15.30 for electrolytic.

Tin.—Although the foreign market was subject to wide fluctuations in the latter part of March and the first part of April, the publication of the March statistics produced an optimistic tone in the market, and prices have become firm at about 48½ cents, New York.

Lead.—The foreign market has been very firm on account of conditions reported from Mexico and Australia, but the domestic market is quiet and unchanged at 4.30@4.35 cents, New York, and 4.15@4.20 cents, St. Louis.

Spelter.—Sales have been effected only at concessions, and the market has declined. Production has been heavy, and plenty of metal is available for sale. A large tonnage has been sold. The last quotations are 5.70@5.80 cents, New York, and 5.55@5.65 cents, St. Louis.

Iron and Steel Market.

The iron and steel market lost tone quite decidedly in April. Conditions had been practically stationary for an unusually long time. Prices of finished steel products had ceased to advance early in October and after a gradual speeding up the mills reached a state of operating at full capacity at about the same time. The placing of contracts for forward delivery reached a high point in October, dwindling thereafter, while during the first three months of this year contracting was uniformly light. In the late months of last year specifications against contracts were extremely heavy, greatly exceeding the shipments, while during the first three months of this year they ran quite uniformly at a rate approximately equalling the shipments.

Thus since October as to prices and the productive rate, and since the beginning of the year as to contracting and specifying, the steel situation had shown no change. In April a definite departure occurred, but the departure was of such a nature that we describe it as merely a loss in ton. Production continued at the full rate. Prices did not recede. Specifications decreased, falling definitely, though but slightly, below the current shipments. Contracting for forward delivery ceased almost entirely.

The mills had a very large amount of business on books, in the form of actual specifications, and the decreased market activity of April did not impair their position, as to the rate of production or as to prices. Continued for two months longer,

the condition probably would do so. In some finished products the mills are booked far beyond midsummer, but without an improvement over April conditions the situation would be impaired by July 1 in some directions.

Whether this relative dullness will continue for a considerable time is naturally dependent upon the causes which produced the condition. In the trade, the common theory is that the tariff discussion is responsible, and if so a definite improvement could hardly be expected for several months. A closer analysis would suggest that there is at least one other cause, that being a psychological one, but one nevertheless susceptible of analysis. In the long run materials make the market, but the market as one finds it from week to week is made by men, not by materials. It proceeds by great movements, for a period upward and for a period downward. The upward movement occurred last year. Then came an anomaly; for six months there was substantially no change in prices, and very little change in general conditions. So long a period of unchanged prices was abnormal, a definite change being overdue. There was not enough latent strength in the situation to produce a fresh upward movement, hence it was natural that the market should lose tone instead. The upward movements are assisted by the price advances, which encourage buyers to contract ahead; with unchanged prices they are not so moved.

While the market has lost ground, it stands in no great danger. The mills are well booked ahead, as to the total tonnage, though here and there are mills with only a little tonnage. There is no danger of cancellations, for nothing new has occurred. A year ago well posted men were convinced that the Democratic party would be successful, and that it would revise the tariff this year, substantially along the lines now proposed. If there is no material improvement from the market condition of April, a slight decrease in production will occur during the midsummer months, but a relatively high rate of activity is assured until almost the close of the year. The serious doubt is as to next year, rather than as to the next six or eight months.

Pig Iron.

The dullness in the pig-iron market continued through April, and was accompanied by somewhat more rapid declines in prices. These declines began late in December, activity having decreased in October and November, but in the first three months of the year the pig-iron market as a whole, averaging all grades and all districts, lost in prices by only about 25 cents per ton per month. In April the losses were at double or triple this rate. The pig-iron decline has been due to psychological rather than material causes. In the first place, producers marked prices up too rapidly, in the late months of last year, having regard to the rate of demand and the large amount of idle capactiy. Production by the merchant furnaces increased rather rapidly from October to February inclusive. Once it became established that prices were not to advance further, buyers adopted their usual waiting attitude, and a period in which buyers bought more than their current consumption required, making purchases for further forward delivery, was succeeded by a period in which they bought practically nothing. The declining prices have confirmed them in their attitude. No new thing occurred in April, for as early as February it became certain that a fresh buying movement would not occur until there had been the usual test of strength, by prices declining to a point which would find some furnaces going out of blast or would find some buyers forced to purchase. That test has not yet occurred, but it will probably occur soon. Production is stationary, consumption is undiminished and if stocks as a whole are increasing they are increasing very slowly, for in some districts, notably the Pittsburgh-valley district, they are lower than for a long time. Prices are down nearly to the cost of production at many furnaces now operating. We quote: Bessemer, $17; basic, $15.75; malleable, $15.50; No. 2 Foundry, $15.25; forge, $14.75, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; foundry iron at Philadelphia, $17.25; Buffalo, $15.50; Cleveland, $16.25; Chicago, $17; Birmingham, $12.50.

Steel.

The floods of late March in the Central West caused a total loss in steel output of more than 250,000 tons, which fell largely in mills producing billets and sheet bars, as a result of which the supply in April was poorer than ever. Prices are nominally unchanged at $29 for billets and $30 for sheet bars and rods, at maker's mill, Pittsburgh or Youngstown. In resuming after the floods the blast furnaces led the steel works, so that the steel works are in comfortable position as to pig iron. About one-half of all the sheet and tin mills were closed on account of flood, and their resumption was retarded by poor deliveries of steel. A Youngstown interest has sold several lots of sheet bars, aggregating 25,000 tons or more, for second half delivery at $27.50, maker's mill, but this seems to exhaust its offerings, and the only market quotable is that for relatively early delivery, given above.

Finished Steel.

Premiums for finished steel products for early delivery are quite unusual, except that plates usually command a premium of $1 to $2 a ton and merchant steel bars bring prices up to 1.75 to 2 cents, depending on quantity. Regular mill prices are given below. Under date of April 12 the National Tube Company issued a new list, advancing steel pipe by about $1 a ton, and the new list was adopted by independents, eliminating the slight concessions which had been given from the old list by several mills. Slight shading in sheets and tin plates disappeared in April, on account of the flood decreasing production, but on galvanized sheets the shading reappeared, to the extent of $2 a ton.

Regular prices at Pittsburgh, unless otherwise stated, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Steel hoops, 1.60 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.67½ cents, Philadelphia; 1.57½ cents, Chicago.

Wire nails, $1.80 per keg, base; plain wire, 1.60 cents.

Sheets, blue annealed, 10-gage, 1.75 cents; black, 28-gage, 2.35 cents; galvanized, 28-gage, 2.40 to 2.50 cents; painted corrugated, 2.55 cents; galvanized corrugated, 3.55 cents.

Tin plates, $3.60 for 100-lb. cokes.

Merchant steel pipe, ¾-in. to 3-in., 79½ per cent off list.

Steel boiler tubes, 3½-in. to 4½-in., 70 per cent off list.

Standard railroad spikes, 1.80 to 1.85 cents.

Button-head structural rivets, 2.10 to 2.20 cents; cone-head boiler rivets, 2.20 to 2.30 cents.

The Western Metallurgical Field

Labor Strike at Globe Smelter

The beginning of April witnessed a labor strike at the Globe plant of the American Smelting & Refining Company, Denver. The Western Federation of Miners induced the men to walk out and demand more pay and better working conditions. The strike continued for a week or ten days, during which time the smelting furnaces were banked or run at such capacity as could be maintained by the technical staff of the plant. The smelting company refused to treat with the Western Federation, but offered to listen to a delegation of the strikers. After being out about ten days, the strikers returned to work, and the plant is again running at usual capacity.

An investigation of the conditions at the smelter indicated that they were not as bad as claimed by the Federation, neither were wages as low as stated. Another rumor and press report, which proved to be untrue, was to the effect that the strikers who returned to work did so at reduced wages.

Prospecting for Potash in Death Valley, Cal.

Preliminary results of the Geological Survey's drilling in Death Valley, California, have now been reported, and the following brief summary is issued:

Reports state that practically the entire area in Death Valley has lately been located as "potash" claims, presumably as association placers, but in blocks of very large area. It is said that a single tract of 17,120 acres, known as the Kali property, was located in May to July, 1912, on the lowest part of the valley. Other groups both north and south of this tract have subsequently been taken up. A vast amount of saline matter is accumulated on and beneath the floor of Death Valley, and it is logical that this area should have been one of the first to attract attention in the search for soluble potash salts.

Four wells have recently been drilled by the Geological Survey to test the general character of the deposits that lie beneath the floor of this valley. Three of these wells were placed in the area of smooth salt in the lowest depression of the valley. The fourth was located in the area of rough salts about 20 miles north of the others. The records of all of these holes are very much alike. These borings are believed to be the first information that has been obtained concerning the character of the deposits underlying the floor of this valley.

Eighty-three samples of muds, salt crystals, and waters or brines have been preserved from these borings for further chemical examination. The preliminary results, given in the accompanying table, include only potash determinations made on natural brines collected from the beds that yielded the principal flows. The analyses are by W. B. Hicks and R. K. Bailey.

Description of Sample	Depth in Feet at Which Sample Was Taken	Total Salts (Ignited Residue) Percentage of Original Sample	Potash (K_2O) in Total Salts Expressed as Percentage of Ignited Residue	Potassium Expressed as Percentage of KCl in Original Solution
Ground water in the salt crust at the "sink"	.5	28.19	3.43	1.53
Water in open "pothole"	.5	27.47	1.20	.52
Water in open "pothole" (same)	9.5	27.48	1.18	.51
U. S. G. S. well, No. 1	6.0	27.87	2.85	1.27
U. S. G. S. well, No. 1	24.0	28.64	2.22	1.01
U. S. G. S. well, No. 1	29.0	28.96	2.35	1.09
U. S. G. S. well, No. 1	52.0	28.66	2.01	.91
U. S. G. S. well, No. 2	32.0	28.33	1.54	.69
U. S. G. S. well, No. 2	38.0	29.16	1.78	.82
U. S. G. S. well, No. 2	70.0	29.96	2.48	1.18
U. S. G. S. well, No. 3	1.0	27.78	2.05	.90
U. S. G. S. well, No. 3	30.0	27.91	1.68	.74
U. S. G. S. well, No. 4	32.0	28.77	2.23	1.02
U. S. G. S. well, No. 4	38.0	28.73	2.12	.97
Average		28.42	2.08	.94

A vast amount of saline material is accumulated in the bottom of this valley, but the mode of its deposition probably is not favorable to selective crystallization on a large scale. Segregation of potash or any other portion of the soluble constituents of the waters may have taken place to a slight extent in the individual salt crust layers, but under the conditions described any such differentiation is likely to have been restricted to the individual layers as units, and therefore has occurred on a scale so small as to be of doubtful practical importance. It seems evident that unless a vast body of saline material has been deposited at one time during a single period of desiccation that there would be little chance for the various dissolved constituents to become segregated one from another on a large scale. There is no record of the drying up of a single large lake of saline waters in Death Valley. Although it is possible that the shores of such a lake might have been completely buried, the assumption that this may have happened must be purely a matter of speculation. Similar reasoning may apply to many other areas in the desert basin region. Great interior lakes have existed in certain areas and may have dried up under such conditions that the salts they contained were deposited in a great body and that the potash and other minerals in the waters were to a certain extent segregated in some portion of the deposits by crystallization. Searles Lake contains a deposit of salt evidently produced in this way, and others may be found, although they are not now exposed at the surface. It does not appear that there is much theoretical justification for the belief that such deposits must be present in Death Valley. It is, of course, possible that the present conclusions are based on insufficient negative evidence, and for this reason any further drilling that may be carried on by private claimants in the Death Valley region should afford a record of much importance and interest in this connection.

Tendencies in Concentration

In a paper recently read before the Colorado Scientific Society, Mr. Edward S. Wiard, of Denver, gave the following classification of concentrating ores, based on the degree of crushing required to unlock their constituent minerals:

Class A. Ores yielding on a coarse initial break, clean concentrate and clean tailing for the large size (between 1¼ in. and 10-mesh), and similar products in the sand and slime sizes.

Class B. Ores yielding on a coarse initial break, large size clean concentrate, large size clean tailing and large size middling.

Class C. Ores yielding on a coarse initial break, large size clean concentrate and a corresponding middling.

Class D. Ores yielding on a coarse initial break, large size clean tailing and a corresponding middling.

Class E. Ores yielding on a coarse initial break, nothing but large size middling; and as a sub-class, ore yielding two or more grades of large size middlings.

The definition of the first three classes indicates a brief outline or flow-sheet for the treatment of a majority of ores. Examples of Class A ores are very rare; but recently a good example has come to light in an ore containing 13 oz. silver per ton, 12 per cent lead and 32 per cent zinc. On crushing coarsely to a limiting size of about 10 mm., screening and classifying, the screen sizes were jigged and the classified products treated on tables. The jigs yielded clean lead concentrate of 63 per cent lead and zinc concentrate of 48 per cent zinc. The jig tailing contained only 1 per cent lead and 6 per cent zinc. Similar products were obtained from the tables, the slimy discharge from which contained only 1.2 per cent lead and 13 per cent zinc, amounting to only 3 per cent of the total mill feed. The approximate percentage saving for the whole operation was 88 per cent of the lead, 90 per cent of the zinc and 85 per cent of the silver.

In large size mills, say above 100 tons capacity, there is a tendency to abandon the familiar trommel line in favor of one or more sets of splitting screens which divide the feed into oversize and undersize for direct feed to jigs. The reason for this trend is found in the excessive cost of maintenance of trommel lines, the time lost in repairs, and the inefficiency of the device under large loads, with the accompanying tendency for the feed to build up at certain points in the system.

In jigging practice the criterion has been the production of concentrate, and it has been commonly believed that unless a shipping product could be made, jigging was not profitable. It is possible, however, by jigging, to produce an enriched portion for subsequent fine regrinding without very much loss, and the enriched portion could be concentrated to the same tailing that would obtain if all the ore had been ground fine before separation. Jigging in such a case would give a greater total saving, and the required equipment would be less.

Company Reports

The American Smelting & Refining Company's fourteenth annual report, for the calendar year 1912, contains little or no matter of general interest. From the financial figures submitted it appears that the company had a profitable year, but the balance of the report lacks detail and definiteness. As an incentive to good service on the part of employees, the company appropriated the sum of $367,822 for distribution as a cash bonus. A new pension system has been adopted whereby male employees who have reached the age of sixty-five years and female employees who have reached the age of sixty years, and who have been continuously in the employ of the company, or of acquired companies, for twenty years, may be retired. The

amount paid to pensioners will be based on an allowance of 1 per cent for each year of service of the average annual pay during the ten years next preceding retirement, but no pension will exceed $200 per month or be less than $20 per month. A sum of $500,000 will be invested to meet the requirements of this system.

The Mount Morgan Gold Mining Company, Ltd., Queensland, Australia, has made public the report of the general manager, Mr. Benjamin Magnus, for the half year ended Nov. 30, 1912. The operations in the mine were very satisfactory and were attended by a gradual reduction in cost. The copper smelter treated the following tonnage of ore:

	Tons treated	Copper produced tons	Gold produced oz.
Copper ore	93,783	3,276	42,744
Gold ore	19,895	530	18,831
Many Peaks ore	54,394	894	413
Purchased ore	178	23	9
Miscellaneous	4,173	282	555
Total	172,423	5,005	62,552

The permanent closing of the Mundic works has caused the smelter to handle the silicious ore formerly treated there. This has increased the cost of producing copper, but the increased gold value of the blister copper gives greater net returns on the operation. A new smelting plant has been started, entirely independent of the old plant, and will embody the latest ideas in modern copper smelting. This new plant is warranted not only on account of the increased net return which will result, but also by the life of the mine. In the meantime improvements are being made at the old plant so that costs there will be cut down as far as possible. Mr. Robert Sticht is assisting Mr. Magnus in the design of the new plant.

The Goldfield Consolidated Mines Company, Goldfield, Nev., effected material reduction in costs during the year 1912, according to the sixth annual report of that company. The ore produced from the mine had an average value of $19.77 per ton, while the average gross realization from ore treatment was $18.40 per ton. Costs were reduced by $1.36 per ton, the noteworthy items of reduction being, milling, 28 cents per ton; marketing concentrates, 61 cents; marketing bullion, 8 cents; general expense, 10 cents. The total expense of milling was $1.61 per ton; concentrate treatment, $0.38; marketing bullion, $0.07. The net realization from operations was $11.75 per ton. This amounted to 59.44 per cent of total production. The operation of the concentrate treatment plant increased net profit from milling operations approximately 50 cents per ton of ore milled.

The Miami Copper Company's annual report for 1912 states that the sixth and last section of the mill was started about the last of February and that the average tonnage treated per section per day was 489 tons of ore. The ore has proved hard to crush, and rigid rolls have been replaced by heavy spring rolls, and the Chilean mills by Hardinge pebble mills. Two important experiments were conducted during the year. The first showed that no greater extraction or reduced cost could be obtained by using the so-called "roughing" process. The second, which is still in operation, indicates that a retreatment process, consisting in regrinding the coarsest sands from the sand-floor reject and middling, followed by classification and table treatment, will give an increased saving of about 1½ lb. copper per ton of ore. The net smelter returns of refined copper amounted to 32,832,609 lb. Based on this return, the cost of refined copper in concentrate on board cars at Miami, was $0.06477 per lb.

The Old Dominion Copper Mining & Smelting Company has issued its annual report for 1912. The following items relate to metallurgical operations. Concentrating ore treated in 1912, was 166,870 tons containing an average of 3.37 per cent. copper, as compared with 140,230 tons of 3.99 per cent grade in 1911. The costs for 1912 were, nevertheless, lower than for 1911, being $0.753 and $1.012 per ton, respectively. Smelting costs for 1912 were $2.304 per ton of new charge smelted, as compared with a similar cost of $2.568 in 1911. The cost of converting per ton of fine copper was $9.45 in 1912, and $8.26 in 1911. Included in the 1912 cost is an item of $13,500 for improvements. The old practice of carrying converter fumes directly into the furnace flue-dust chamber was abandoned, and a new flue-dust chamber was erected for converter gases. In six months the recovery of dust has amounted to 34,000 lb., averaging 45 per cent copper. Over 87 per cent of this was finer than 200-mesh, and would have been lost under the old system. The first unit of a basic converting plant has been installed, being of the Great Falls type, and it is expected that converting costs for 1913 will be materially reduced.

The fifth quarterly report of the *Chino Copper Company*, covering the fourth quarter of 1912, shows a total production of copper for the year of 29,237,966 pounds. Each succeeding quarter of the year showed an increase in tonnage treated and copper produced. During the fourth quarter the quantity milled was 452,650 tons of ore, averaging 1.844 per cent copper, yielding 10,781,621 lb. copper. The grade of ore was lower than for the preceding quarter when it averaged 2.33 per cent copper. The total net profits from all sources for the year was $2,352,822. The earnings for the fourth quarter were calculated on an average price of 14.532 cents per pound of copper. The cost per pound of net copper produced, after making smelting deductions, was 8.49 cents per pound, as compared with 6.57 cents for the preceding quarter. This increased cost is due partly to the ore treated, which contains a large percentage of iron, thus giving a lower ratio of concentration and lower grade of concentrate. The ratio of concentration was 15.14:1, the concentrate averaging 18.03 per cent copper as compared with an average of 25.02 per cent for the preceding quarter. The average extraction for the quarter was 64.58 per cent, compared with 65.15 per cent in the preceding quarter. The fifth section of the mill was put in operation November 28, 1912, but practically only four sections have been in operation during the fourth quarter on account of making improvements.

The Ray Consolidated Copper Company's report for the fourth quarter of 1912 shows that the gross production for the year was 35,861,496 lb. copper. The average grade of ore treated during the fourth quarter was 1.6737 per cent copper, which is about the average for the year. The extraction was 68.102 per cent, compared with an extraction of 68.2783 per cent for the whole year. The average cost per pound of net copper produced during the quarter was 9.828 cents, as against a cost of 10.047 cents for the first three quarters. During the quarter six sections of the mill at Hayden were in operation. The seventh section is now available, and the eighth is nearing completion.

In the annual report of the *Arizona Copper Company* for the year ended September 30, 1912, the general manager gives the following data regarding the operation of mill and smelter. During the year 782,100 tons of ore were concentrated, the ratio being 6.74 tons into 1. Improvements in the concentrating plant raised the daily tonnage from 907 tons per running day in the previous year to 1573 tons last year. Concentrate re-dressing plants were installed at both mills last year, for the purpose of making a higher grade, cleaner product. At the oxide ore concentrator and leaching plant 108,230 tons of ore were treated, yielding 89,800 tons of tailings which were treated in the leaching plant. This treatment yielded 9.71 per cent of the total copper production.

In the smelting department the average yield from ores and concentrates was 12.03 per cent copper. The elimination of silicious matter in the re-dressing plant at the concentrator reduced the amount of flux required, increased the capacity of the furnaces and effected an economy in smelting. In the converter department one converter was run with basic lining, the operation being so successful that the remaining converters have been changed to conform to that practice.

The erection of the new smelting works is progressing rapidly, and it is expected that operation will commence in late summer or early autumn. The prospective date of completion and operation is considerably earlier than expected.

Evolution in Methods of Handling Slime—II

By H. N. Spicer

New Zealand is an interesting country from a metallurgical point of view, particularly on account of its contribution of the tall, pneumatic-slime-agitation tank to the cyanide industry. This device is the invention of F. C. Brown and S. D. McMiken, of the Komata Reefs mill, where the first of these tanks was built and used for the agitation of ore slime. Owing to the fact that this tank was first adopted in North America at Pachuca, Mexico, it has come to be more commonly known as the Pachuca agitator, thus in a way detracting from the credit due to its inventors and the country of its origin.

It will be apparent from the following notes that the methods of handling slime in New Zealand are much more advanced than those described in the preceding paper on India, in which country the crudest methods have prevailed up to the present time. As in other countries, certain typical features have been developed in New Zealand cyanide practice, which are common to all the mills and distinguish them from others.

FIG. 1.—WAIHI MILL, WAIHI GOLD MINING COMPANY

The Waihi Mills

The Waihi mine is the largest in New Zealand, and the ore is treated in two large mills. The Waihi, of 90 stamps, and the Waihino, of 200 stamps. At both mills the ore is crushed in water, thus introducing the necessity of dewatering before cyaniding.

At the Waihi mill, shown in Fig. 1, the stamp pulp, containing from 7 to 9 tons of water per ton of ore, is elevated by wheels to spitzkasten, 4 ft. square and 4 ft. deep, two of these being arranged in series for each twenty stamps. The separated sand is reground in tube mills, and after amalgamation and concentration, is classified in other spitzkasten. The sand pulp is run to circular collecting vats previously filled with water. Any slime accompanying the sand flows out of the vats and is returned to the main separating cone. When the sand vats are full they are allowed to drain, after which the sand is leached with cyanide solution.

The slime, 90 per cent to 95 per cent of which will pass a 200-mesh screen, flows to a set of dewatering cones, 5 ft. square and 5 ft. deep, in which the pulp is thickened to a consistency of three or four parts of water to one of solids. This thickened slime is further dewatered in a device peculiar to this plant. It consists of three flat-bottom settling tanks, 32 ft. diameter and 18 ft. deep, provided with annular overflow launders which carry off the clear water. In the center of each tank is a vertical shaft with a heavy wooden arm attached at the bottom, the whole being slowly revolved by an overhead gear. Near the periphery of the bottom of the tank is a 4-in. valve which is opened at intervals of half an hour for the discharge of accumulated thick slime. This product contains about 1½ parts of water to 1 of solids. It is then mixed with cyanide solution and pumped to Pachuca agitators.

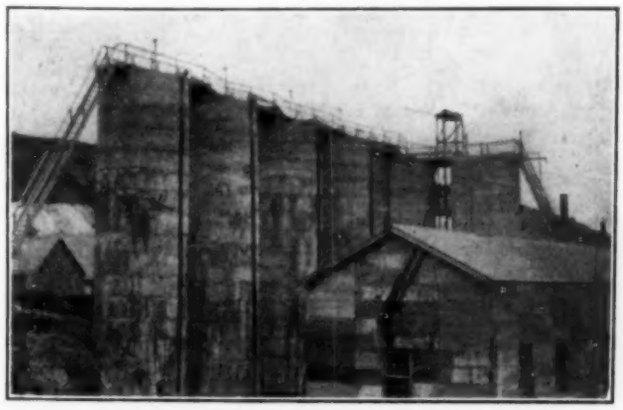


FIG. 2.—REINFORCED CONCRETE PACHUCA TANKS, WAIHI MILL

These agitators are worth more than passing mention, as they are built of reinforced concrete, 55 ft. high and 13 ft. diameter, inside measurements. The walls are 5 in. thick; but similar tanks at another mill have been built with only 4-in. walls, and have proved satisfactory. The reinforcement is old rails and wire rope, the latter being woven horizontally through the former which stand vertical. The cost of these tanks was appreciably lower than the bids for similar steel tanks, indicating that under certain conditions where cement and labor are comparatively cheap, it may prove profitable to build concrete agitators. A closer view of the agitators is given in Fig. 2.

A different procedure obtains at the Waihino mill of the same company, shown in Fig. 3. Sand and slime are separated in spitzkasten, the former being shovelled into filter-bottom concrete tanks and leached with cyanide solution. The slime from the dewatering cones, containing about 1½ parts water to 1 of solids, is further dewatered by vacuum filters which reduce the moisture to about 30 per cent. These filters are of the stationary Moore type. After a cake has been formed, the vacuum is cut off and the slime allowed to fall back into the tank; if necessary, compressed air is blown into the leaves to aid in dislodging the cake. The tank has a V bottom, terminating in a goose-neck through which the sludge is periodically withdrawn into a mixer where cyanide solution is added.

The slime is then elevated to steel Pachuca tanks by means of two-stage tailing wheels. The agitators are without the customary central lift-pipe, but are found to give good results. The pulp is agitated for about 7 days, with air at 40 lb. pressure. The great length of time required for agitation is mainly to effect solution of the silver.

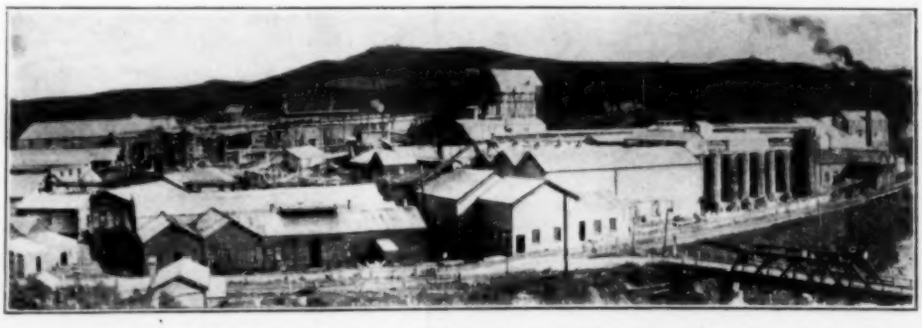


FIG. 3.—WAIHINO MILL, WAIHI GOLD MINING COMPANY

Moore type vacuum filters are used at both these mills. Their construction suggests an interesting departure from the usual custom, corrugated steel sheets being substituted for the ordinary cocoa-matting center.

Concentrates from these mills are treated by cyanidation after being reground impalpably fine. The pulp is agitated for fourteen days in high, narrow tanks, and then filtered in plate and frame presses. The extraction is about 94 per cent.

Waihi Grand Junction Mill

The present practice at this mill represents an interesting development from the separate treatment of concentrate, sand

FIG. 4.—WAIHI GRAND JUNCTION MILL

and slime, to a system in which all the ore is treated as slime. With this development came the adoption of crushing in cyanide solution, this being one of the few mills in New Zealand to adopt this practice. Preliminary dewatering of the slime is thereby avoided, and the time of treatment is shortened by the more rapid solution of the precious metals during the crushing and grinding process. The mill is shown in Fig. 4, at the extreme right of which can be seen the tall agitators.

The ore treated here is practically the same as at the Waihi mill, and the process consists in crushing by stamps, separating sand from slime in cones, regrinding the sand in tube mills which work in a closed circuit with cones, settling and agitating the slime, and filtering by vacuum. The underflow of the cones is run over the Wilfley tables formerly used as concen-

FIG. 5.—TALISMAN MILL, KANGAHAKE, NEW ZEALAND

trators when concentrate was treated separately, but now used merely as classifiers to separate some fine sand for return to the tube mills.

The slime separated by the cones contains about 10 parts cyanide solution to one of solids, about 95 per cent of the latter being finer than 200-mesh. It is settled in cone-bottom tanks, 22 ft. diameter, the clear overflow from which is returned into the mill system. The thickened pulp is drawn continuously from the bottom of these settlers, having a consistency of about 1½ parts solution to 1 of solids, and is pumped to a series of flat-bottomed storage agitators fitted with stirring arms revolving at 4 r.p.m. From these the pulp is pumped to the high agitators, which are without the usual central lift-pipe. Air is used at 38 lb. pressure, and the time of agitation is 18 hours, the latter having been reduced from a period of several days. This is an improvement which may be ascribed, in part, to the practice of crushing in solution. Vacuum filtration in a Moore filter completes the slime treatment.

The Talisman Mill

The principal features in handling slime at this mill are the use of diaphragm cones for classification, and huge spitzkasten for dewatering. The latter may be seen in Fig. 5, at the top of the lower building. The ore is stamped in water, and classified into sand and slime in diaphragm cones, the diaphragm being near the bottom, with a ½-in. space all around. These discharge a thick pulp, about one to one, which is ground in tube mills, combined with the overflow of the cones, and again classified in a second set of diaphragm cones, the underflow of which is returned to the tubes. The overflow from the second set of cones is amalgamated, whereby from 30 per cent to 35 per cent of the valuable metals is recovered. Concentration on vanners is the next step, the concentrate being shipped to smelters.

The combined vanner tailings run into steel vats, using Butters and Meins distributers. The vats are originally filled with water through which the sand settles, while the slime overflows.

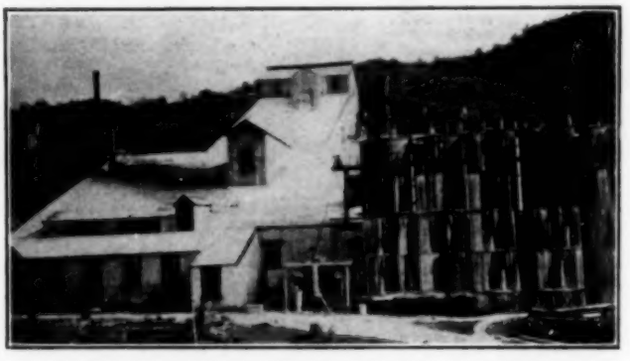


FIG. 6.—KOMATA REEFS MILL

When a vat is full of sand it is drained and leached with cyanide solution. The slime is pumped to the large spitzkasten previously referred to, there being two sets of nine pyramidal boxes, each 6 ft. square and 6 ft. deep. Limewater is added to facilitate settling. The underflow from these spitzkasten passes to rectangular settling tanks, 20 ft. by 10 ft. area and 7 ft. deep, each having two pyramidal hoppers in the bottom, 10 ft. square and 7 ft. deep. The overflow from these passes in succession to two sets of flat-bottom circular vats to effect further settlement before flowing to waste.

Slime for agitation, consisting of water and ore in the ratio of 2:1, is drawn from the settling vats and pumped to tall agitators which, like those at the Waihi, have no central lift-pipe. When an agitator tank is filled within 18 in. of the top, the flow is diverted and the slime allowed to settle for an hour or more. The clear water is decanted, cyanide solution added to form a pulp of one part solids and one part solution, and agitation proceeds for 24 hours. Air is used at 40 lb. pressure. Filtration in a Moore fixed-frame type of filter completes the treatment of slime at this mill. At the present time the Talisman company has a new mill under construction, the operating costs of which are expected to show a considerable reduction in comparison with those of the mill described above.

Komata Reefs Mill

The view of the Komata Reefs mill, Fig. 6, is interesting as showing the first tall air-agitation tanks ever erected for the treatment of ore-slime. These are the two at the left, behind which are two others all resting on the elevated platform

The other tanks at the right, with continuous sides resting on concrete foundations, were erected later. These agitators are equipped with central lift-pipes, which are an essential part of the original invention of Brown and McMiken. Another important contribution to the mechanics of cyanidation originated with Mr. Brown, namely, the iron plate-and-rib liner for tube mills.

The practice at this mill conforms in some respects more nearly to modern ideas about handling slime. The ore is crushed in water and classified in spitzkasten, the sand being reground in tube mills. The tube mill discharge is classified in other spitzkasten, the sand passing to leaching vats, and the slime to Dorr continuous thickeners. The latter continuously discharge a pulp containing less than 40 per cent moisture, which is pumped to the agitators and mixed with cyanide solution to a consistency of about 65 per cent solution and 35 per cent solids. Intermittent charges are agitated for seven or eight days, after which the pulp is drawn off to a Dorr thickener which removes clear cyanide solution and gives a pulp containing about 45 per cent solution, which is finally displaced in a Moore "turnover" type vacuum filter.

Waihi-Paeroa Tailing Mill

The practice at this mill is a good example of what can be accomplished in re-treating tailings from other mills when favorable conditions exist. The various mills at Waihi, Karangahake and Waitkauri discharge tailings into the Ohinemuri River, which has a fall of about 20 ft. per mile to a point about 6 miles above its junction with the Waihi River. Here is a stretch of more nearly level country where the stream deposits its burden of sand, while the slime is carried on to sea. It has been estimated that the mills on the upper reaches of the river have crushed from 5,000,000 to 6,000,000 tons of ore, and have discharged this quantity of tailings into the stream. The sand has accumulated in the natural settling basin referred to; and an estimate of the quantity of such material deposited is about 2,000,000 tons. To this must be added the current annual contribution of the upper mills, amounting to about 500,000 tons of sand and slime.

The project of the Waihi-Paeroa Gold Extraction Company, Ltd., involved dredging this sand from the river bed, grinding in tube mills in cyanide solution, thickening, agitating and filtering the slime thus produced. This has been successfully carried out in a 500-ton mill.

The dredging machinery, consisting of oil engine, air compressor, centrifugal pump and air-lift, is mounted on a pontoon. The lift-pipe is telescopic, and can be lowered to any desired depth. The centrifugal pump is used to force water through a series of jets near the bottom of the air-lift to thin the sand to the desired consistency for pumping by compressed air. The sand is pumped into barges and towed to the wharf.

Charcoal and river sand are removed by treatment on vanners, after which the valuable residue is ground in tube mills working in a closed circuit with classifiers. The tube mill product is all finer than 150-mesh. This is elevated by two-stage tailing wheels to large settling cones from which the thickened pulp is discharged at a consistency of about 1½ to 1. It is pumped to tall air-agitators which are without central lift-pipes, and agitated for forty-eight hours.

The Moore type of vacuum filter is used, the tanks being circular and cone-bottomed. One tank is used for loading and one on either side of this for washing and discharging respectively. A disintegrator is placed in the bottom of the discharging tank, for the purpose of churning the caked slime with water before finally discharging it to waste.

Blackwater Mines Reduction Works

Stamping in cyanide solution is practiced at this mill, which is illustrated in Fig. 7. The view shows plainly the tall agitation tanks, dewatering cones, Moore filter and sand vats. The

FIG. 7.—CYANIDE MILL, BLACK WATER MINES

dewatering cones and Moore filter tanks are of wood; the agitators have no central lift-pipe.

The first separation of the stamp pulp into sand and slime is made in the leaching vats, the overflowing slime and solution being pumped back to a series of large wooden dewatering cones. The latter thicken the pulp to a consistency of about two to one, after which it is agitated. Vacuum filtration completes the slime treatment.

Summary

The characteristic features of New Zealand practice in handling slime may be summed up as follows: Crushing in water prevails at most of the mills. Intermittent or charge agitation is customary, the continuous method having been tried and abandoned on account of the tendency of the coarse solids to build up in the tanks. All New Zealand mills use the tall air-agitator of Brown and McMiken, but without the central lift-pipe, except at the Komata Reefs mill, where the agitator was invented and first used. It is generally claimed that the use of the central tubes give no better results; and as this apparently the only feature of the agitator covered by patent it is generally omitted. High air pressure, 40 lb. per square inch, is customary, being about double that used in America.

Tailing wheels still hold their own as elevators in New Zealand mills. The last to be erected are at the Waihi-Paeroa, where a two-stage lift is adopted. The Brown ribbed liner for tube mills, which is being more generally adopted in other countries, was originated in New Zealand, and is generally used there. The Moore filter has a strong hold, all the mills of any consequence using it. One of the peculiarities of construction of filter leaves is the substitution of corrugated sheet steel for cocoa matting in the center. It is claimed that this type of leaf filters just as well as the other, is lighter and dries more quickly; also that it is more readily cleaned if leaks occur in the outer canvas.

Denver, Col.

Carbon Electrodes for Electrolytic Cells

By John Härdén

A good deal of artificial carbon electrodes are to-day used for electrolytic work, and the subject of their manufacture is still rather important, although it may perhaps be right to say that the bulk of carbons made at present and more so in the future are used for electrothermal purposes. A few data regarding the "electrolytic carbons" may, therefore, be of some interest.

In the United States the graphitized electrode, first introduced on a larger scale by Dr. Acheson, whose name has rightly become famous in connection with this industry, has gained a well-deserved market. In Europe these electrodes have also obtained due recognition, but here the conditions are considerably different so that common amorphous electrodes are still to a considerable extent in use.

This may be explained by relating the writer's experience in some electrolytic works in Russia some years ago. These works produced on a large scale chlorine and caustic soda from ordinary sodium chloride obtained from the Black Sea and from the Caspian. Having been engaged previously as head of the testing department of a large carbon electrode factory, the writer was given ample opportunity to compare and to judge the particular "pro's and contra's" in question.

Since a heavy duty was levied on the finished article, it was found uneconomical to import the electrodes from abroad, and no factory was in existence in the country itself. The preparing of the raw material, however, is of great importance, as will be seen later. Therefore the ready prepared mass was obtained from a foreign carbon factory on the other side of the frontier, transshipped as raw material, consequently free of duty, and pressed and calcined into proper electrodes at the works. This realized a considerable saving and made the large outlay of a complete electrode plant with expensive machinery and large stock of materials unnecessary.

In spite of this, the cost of electrodes was a heavy toll on the works' budget. The cells consisted of an earthenware top in which seven round electrodes, about 4 inches in diameter by 2 feet 6 inches in length, were fastened. Diaphragms of asbestos separated the two compartments, anode and cathode room, and as the electrodes were worn off, the whole set of cells, six in each batch, had to be dismantled when repairs were to be made.

In order to lengthen the life of the cell, some graphitized electrodes were tried, with partial success as was expected. In those days, however, such electrodes imported from U. S. A. became very expensive, not only on account of their initial considerably higher price, but the exorbitant freight and unavoidable loss in breakage. It was generally found that the outlay for such electrodes, under the prevailing circumstances, was not in any way justified by their longer life, since the local labor was so inexpensive that it did not come into play, and the saving in material, although decidedly marked was not sufficient to compensate the other outlay.

This state of affairs is no doubt responsible for the fact that non-graphitized are still being used to a great extent, notwithstanding that the graphitized material is superior as regards actual wear and tear. In order, however, to test the matter more fully, the firm decided to graphitize its own electrodes on the spot, thereby utilizing a method worked out somewhat earlier by the author, and which differed in various respects from the Niagara practice. Some of its points have been briefly described in an article in METALLURGICAL AND CHEMICAL ENGINEERING, vol. IX. p. 130, March, 1911, headed "Some Electric Furnace Notes." A furnace was built and put into operation and after a few failures and alterations we succeeded in obtaining a continuous working, producing some 12 to 16 graphitized electrodes per day.

But even then, the economy was very problematic. Although the cost of power was not excessive, since it was produced by means of cheap local coal, the saving generally was not so great as at first expected. The electrodes prepared in this way, though fully graphitized, showed a marked tendency to scale off in the bath, an experience, by the way, which has been since made also in electric furnace work, when the higher conductivity of the graphitized carbon has been made use of in order to increase the current density, thus decreasing the running costs of electrodes generally.

The graphite scales thrown off lodged in the bottom of the cell and clogged this up to such an extent, that it had to be opened and cleaned long before the electrodes were worn out, thus rendering the apparent saving illusive. This was due, to a great extent, probably to the fact that the material from which the electrodes were made was not quite suitable for this particular purpose; if a carbon factory proper had been connected with the works, it is quite possible that the fault may have been remedied to a greater extent. This problem of scaling off is one which the manufacturers of graphitized carbons should bear in mind; a material which, when graphitized may serve very well for one purpose may be quite unsuitable for another; in this case probably because every trace of the carbonaceous binder was thoroughly burnt out and converted into graphite as well as the main body, a fact which was fully shown by means of microphotographs, taken by the author and published in *Zeit. f. Electrochem.*, 1901, p. 324, and also by Sproesster in No. 71 of the same journal.

Large grains with adequate binding material, pressed under moderate force, which gives a very good result for furnace electrodes when used as amorphous electrodes, showed this scaling off in a marked degree when graphitized, although the porosity generally is much lessened. Finer grain, with less binder, stamped hard, calcined and graphitized, showed a better result in this respect, though the resistancy against the chemical action of the electrolyte is then generally lessened.

For the same reason it was also found that material which had been previously graphitized and then ground to suitable size, gave with an adequate binder of tar and high pressure as well as good calcination a comparatively better result. The economy of this working, however, was less satisfactory.

The Testing of the Electrodes for Electrolysis

The testing of the electrodes is of great importance, both for the manufacturer and the user. For the former, as it will tell him about the raw material and its preparation and also the subsequent manufacture so as to avoid complaints, etc.; for the latter to make sure that he has the right kind of material, since a fault may cost him a good deal if it is found out only when the cells have been built in. Generally, it should be emphasized that an electrode should be tested in the solution and under conditions as near as possible to those in actual use.

Preliminary, the following observations should be made:

(1) Homogeneity and grain.
(2) Absence of cracks and flaws.
(3) Porosity and specific weight.
(4) Contents of inert material.
(5) Electric conductivity, and last but not least—
(6) Resistivity to corrosion.

(1) Homogeneity and grain should give an idea of the uniformity of the raw material; if too fine and of a deep black luster, too much dust and "lampblack" material has been used. Such an electrode is likely to be less strong mechanically and will often cause larger lumps to drop off in the cells. A grey structure with light, shining points or globes indicate the presence of a large quantity of petroleum-coke, which if used in excess may behave similar to the former kind.

As a rule, the structure of a good, durable, *amorphous* electrode should be of even grain, not too fine, the grains being well and evenly united by the binder of a dark greyish color. Each particle of the mass should be thoroughly enveloped in a film of the cement or binder before pressing, and it is this part of the preparation of the raw material which calls for the utmost care and attention, and which, besides the choice of the material and the calcining, is the mainstay of the durability of the electrode.

Ingenious machines, the description of which space forbids.

have been invented and put into use to produce this effect, and money spent on this by the manufacturer is well invested.

(2) The absence of cracks and flaws is of course equally essential for the same reasons. These may be caused (and are usually caused) by inadequate treatment of the raw material, but also through false methods of calcining. The firing usually takes place in a gas-fired "ring" furnace, in which the electrodes are placed, embedded in coke dust, etc. If this packing is uneven, the electrode will "hang" and edge cracks are the result.

Such cracks are bad enough in electrodes for electrolytic work, since the increased resistance of the carbon may cause a poor distribution of the current and also cause the faulty electrode to drop off, but they are distinctly bad in the case of furnace electrodes, where a broken electrode may mean a whole smelt to be spoiled. Such occurrences are now-a-days rather rare, where proper "ring" kilns for calcining are employed, but a less scrupulous foreman may cause faulty products even with the best kilns.

A good, sound amorphous carbon should, when hanging free in a sling, and tapped with a steel bar, give a clear, distinct ring like a bar of hard cast iron, and endure nearly the same rough handling as such. A graphitized carbon again should give a deep "thud," yet clear and sound, be capable of polish by means of a brandishing iron and be soft enough to be cut with a knife, while the amorphous carbon is hard and should not permit cutting with other tools than the chisel and emery wheel.

(3) The porosity and specific weight are also both of importance. A certain amount of porosity is of advantage, as it will increase the active surface of the carbon; but if in excess, the bath liquid will enter the pores and cause a rapid development of gas which tends to break off small chips, especially at the less porous edges and thus shorten mechanically the life of the electrode. A good practice, gained by comparing a new carbon of unknown merit with a test piece of a well-known quality, whereby a medium strong magnifying glass should be used, will be found helpful. The pores should be even and not too marked, resembling those on a hard sugar loaf, the calcined binder causing the surface to appear slightly glazed.

The general way in which the specific gravity is determined may be shortly described.

A piece of the carbon, not too small, preferably cut out to regular size of a few cubic centimeters, is boiled in alcohol, dried in the oven, where it may cool and the weight is taken (a). Its volume (v) is then determined in a measure, the water is wiped off quick and the wet weight (b) is taken. The mean specific gravity is then a/b and the real specific gravity is

$$\frac{a}{v-(b-a)}$$

It may also be determined by means of the method of suitable heavy liquids, such as a mixture of bromoform and chloroform of known specific gravity.[1]

The relative specific gravity varies between 1.4 to 1.6 while the real specific gravity is generally between 1.6 to 1.8, and the porosity $\frac{100\,(a-b)}{a}$ is ranging from 11 to 22.

(4) With inert material is understood such contaminations which do not act as electrode material proper, but may be due to inferior purification of the raw material. This is determined through ash-analysis, i.e., combustion in oxygen and air, whereby both the CO_2 evolved and the remaining ash should be weighed. Good carbons made from petroleum coke and lampblack have usually only about 0.3 to 0.5 per cent of ash, while anthracite electrodes may have 3 to 4 per cent.

Acheson grapihte electrodes show as a rule a very low content of ash, due to the high temperature at which they have been treated. Electrodes for electrolysis ought not to have more than 2.5 to 3 per cent ash, while 4 to 5 per cent, as is sometimes found, is distinctly disadvantageous. The carbons for electric furnace work should suit the conditions for which they are to be used, i.e., for carbide furnaces a content of 3-4 per cent lime is quite tolerable, for steel furnaces 1 to 3 per cent iron, etc., but the life of the electrode will suffer from a higher content of these substances.

The microscopic examination may also be of considerable use for determining the quality with a skilled observer, but it would carry us too far to enter further into this without the reproduction of the various sections. The author has taken a number of microphotographs of carbons manufactured in different ways and he may state as a general rule that the microphotograph will show clearly if rapid consumption of the carbons is due to insufficient calcination of the binder or if the main raw material was of an unsuitable nature. Again in this respect, the graphitized electrode gives a distinctly superior result, as the sections taken prove that practically all of the binder is converted into graphite carbon.

(5) The electric conductivity is naturally of considerable importance, for the more voltage we must apply to a cell, the greater is the dead cost of the electrolysis. That this is a weighty question is obvious, if we remember that a cell is usually operated wtih several thousands of amperes and a slight increase in resistance will soon run up the bill. This is also, though not to such a degree, of importance in furnace work, as may be clear from the following experience of the author.

A set of metallurgical furnaces was operated with carbons of 14 in. diameter at 4000 amp. The stumps were screwed on to the new electrodes by means of threaded plugs and graphite paste. A firm offered a brand of new carbons for which they claimed superior quality at a somewhat lower price than the usual cost of amorphous electrodes. The average resistance of a new carbon not in use was found to be within the proper limits, and a couple of the carbons were put into use as a test.

After a few runs it was found that the carbon scaled off in a remarkable way and one of the electrodes, having been jointed in the meantime, became red hot in the joint, in spite of the fact that the joining was made with special care and with the material specified by the maker. The voltage drop in a 5-ft. unjointed and in a 6-ft. jointed was taken and found to be 4.8 volts in the former and 6.9 volts in the latter at 4000 amp., consequently 19.2 kilowatts were wasted in the former and 27.6 kilowatts in the latter which, of course, made the use of these electrodes an impossibility, since the reduction in initial cost was in no way equal to the loss in power. It may be added that inductance and leakage was duly considered, in any respect it was of little importance, since the periodicity was fairly low, only 22.5 per cent.

The specific resistance is

$$\text{Sp. R.} = \frac{r\,q}{l}$$

where r is the measured resistance of a strip in ohms, q is the cross-section in square millimeters and l the length in meters. It ranges usually for amorphous carbons from 58 to 75, for graphitized electrodes from 6 to 12 and for charcoal electrodes up to 150.

As it is, of course, of great importance that the fastenings of the carbons in an electrolytic cell have as low resistance as possible, it is found advantageous to provide the ends of the carbon slabs with a good coating of graphite paste in all such cases where graphitized carbons cannot be used for one reason or other. The author prefers, where the conditions of the cell permits, to cast a shoe around the upper end of the carbon bar of a type of metal which will contract strongly over the carbon, yet be soft enough to avoid an undue strain on the bar. An alloy of lead, antimony, and bismuth in proportions tested for different brands of carbons has been found very suitable, and as only a thin coating is required, the expense is amply justified. Copper plating is usually found obnoxious, even if the nature of the electrolyte is such as to not corrode the copper, since it is found impossible to eliminate all traces of the acid of the copper bath from the pores of the carbon and the plating will invariably peel off and cause poor contact in time.

[1] *Z. f. El. Chem.*, 1901, p. 973.

(6) The resistivity to corrosion is the vital point in question, as has already been mentioned. If a carbon is used as anode in a cell, the carbon-block is invariably disintegrated, more or less, depending upon the nature of the carbon and also of the solution forming the electrolyte. Some of the products, in the case of an alkali cell are gaseous, some go into solution and the bulk is setting off as a slime. Oxygen-giving solutions are more active in this respect, but chlorine is also an enemy to the electrode.

If amorphous carbons are used, one may frequently find a product formed called mellogen, which may be characterized by the formula $C_{11}H_2O_4$, which is soluble in alkali and in concentrated sulphuric acid. This will be changed into mellithic acid ($C_{12}H_6O_{12}$) at ordinary temperatures. The same may be observed in using retort carbon, while graphitized electrodes[2] will form graphitic acid $C_{11}H_4O_8$.

The most reliable way of testing the carbons in this respect is to demand from the maker two or three pieces cut out from the electrodes he intends to deliver and subject these to electrolytic test under such conditions as those prevailing in an ordinary cell, i.e., in a salt solution for chlorine-alkali electrolysis, for instance.

The author used for factory testing the following arrangement: 6 to 12 cells consisting of alkali—and acid-proof enamelled cast-iron vessels were connected in series, the cathodes being sheet iron bent into suitable shape. The carbon anode was suspended in a chamber made of earthenware, provided with asbestos diaphragm and open at the bottom. The solution was 20 per cent sodium chloride, which was kept constant, both to concentration, volume and temperature. The current density was also held constantly at 0.1 ampere per square centimeter by means of meters and regulating device.

A clock was so arranged that a series magnet would permit the clock to go as long as the cells were in operation, but arrested the work automatically when the current was shut off from the cells; in this way an absolutely reliable time record of the duration of the test was obtainable.

The gases evolved in the anode chamber were sucked through a weighed potassium hydrate solution, in order to determine the quantity of carbonic acid produced.

The carbons were cut into regular shape and size, 3 x 1 x 10 cm, brushed, boiled in water, dried in warm alcohol and hot air, and, after the connection lead had been cast on, carefully weighed. It was then immersed in the cell, a soft rubber packing being used to exclude leakage of gases.

The current was then turned on and allowed to act during a given time, usually 48 hours. The electrode was taken out at regular intervals of 12 hours, boiled in water and dried in alcohol and weighed as before. In this way a curve was plotted, showing the deterioration as it proceeded, and the life of the carbon could in this way be easily predicted, if compared with the curve of a standard electrode of well-known wearing quality.

It was very interesting to observe how in many cases the the curve was at first very flat, showing a slight wear only, but as the test proceeded the curve rose rapidly, owing to the fact that the outer layers were more calcined than the inner, also showing greater porosity, whereby the carbon particles were thrown off more readily. By comparing the voltage required by the cell during the test one could very well foretell when it would be more economical to exchange the carbons, even if they were not worn down totally, similar to the critical point of a carbon filament lamp.

One might object that this test was more destructive than the actual conditions in a regular cell, and therefore not conclusive. This, however, is counteracted by the comparison with the standard carbon, tried before in actual use and from time to time submitted to a blank test in series with the new specimens to be tested. The variations observed were so slight, that during the 3½ years the author used this method for determining the wearing quality of such carbons, the variations could very well be neglected and the test was found sufficiently accurate for decision, both for manufacturing and ordering.

A few figures gathered by the author may serve to give an idea of the losses under above conditions; the figures are given in grams per 10 amperes per hour, 20 per cent salt solution and 0.1 ampere per square centimeter density.

1. Conradty	5.17	
2. Le Carbon	2.80	(Graphitized)
3. Plania works	1.90	(special composition)
4. Schiff & Co. "N"	5.65	
5. Retort Carbon	2.92	
6. Acheson Graphite	1.06	
7. Author's Graphite	2.02	

As will be seen from the above, the graphitized electrode is generally superior to the ordinary carbon, but, as already stated, local conditions at the works may under circumstances be such, that one has to use ungraphitized electrodes on account of costs, etc.

The specimen No. 3 shows that it is also possible to produce an amorphous carbon with a wearing quality practically as good as the graphitized one, but it is only just to add, that, owing to its special nature, the price of this was almost the same as a graphitized one. Apart from this, the specific resistance was about eight times higher, and the material very hard and brittle, wherefore it could not be recommended as a good substitute for the graphitized brand.

It is quite clear from the above, that the judging of the merits of a good carbon for electrolytic work requires a good deal of consideration and patience, if a fully economical working result is to be arrived at. It should be borne in mind that the general points set forth above do not by any means exhaust the subject of the testing, but they will serve to give an idea of the faults and merits to bear in view.

Luton, England.

[2]See Zellner, Künstliche Kohlen.

The Fourth International Congress on School Hygiene will be held in Buffalo, N. Y., from Aug. 25 to 30, 1913, under the patronage of Woodrow Wilson, President of the United States. Delegates will attend from all the leading nations, from every college and university of note in this country, and from various other educational, scientific, medical and hygienic institutions and organizations. The congress is further open to all persons interested in school hygiene. Membership may be secured on the payment of a $5 fee. Applications should be sent to Dr. Thomas A. Storey, College of the City of New York, New York City.

The **American Metal Company,** of New York City, has purchased 2500 acres of Pittsburgh coal lands and 350 of surface land at Burgettstown. On this surface land the company will commence to build a zinc and sulphuric acid plant. A subsidiary company, known as the American Zinc & Chemical Company, is being organized with a Pennsylvania charter to operate the new plant, and Mr. N. L. Heinz, of LaSalle, Ill., one of the best-known zinc and sulphuric acid experts in the United States, will be the general manager and has been made a member of the board of directors. What is known as a one-unit plant will be erected and will have a capacity for producing 40,000 tons of sulphuric acid and 20,000 tons of zinc annually. The plant will be so designed that it can be enlarged rapidly, and it is the intention to build additional units until the annual capacity amounts to three or four times the original.

A shaft will be sunk and coal mining operations will be commenced coincident with the opening of the plant. The initial investment in this new plant will be not less than $2,000,000, and it is expected that when the entire plant is completed and in full operation, which will probably be within three years, it will represent an investment of approximately $5,000,000.

The new plant will give employment to 500 men at the start and as the plant is enlarged this number will be increased until about 1500 men will be employed.

A Modification of the Jäger Method of Gas Analysis

By S. H. Worrell

In the November, 1911, number of METALLURGICAL AND CHEMICAL ENGINEERING, there was an article on a simple method of determining methane in producer gas and enriched water gas. The method was that made use of in ordinary combustion analysis of carbon compounds; namely, the use of copper oxide as a source of oxygen. The method is as old as Fresenius, of course, who absorbed the products of combustion and weighed them. In the article just referred to I made the statement that a fairly thorough search of the chemical journals failed to reveal anything published on the subject. I might have added also that the method was not to be found in any text book on the subject of gas analysis that is at hand in our library. Shortly after the article appeared, in looking up another reference, I found that E. Jäger had made use of the method and published it in *Journal für Gasbeleuchtung* in 1898. However, he went a step further, and by first heating the tube containing copper oxide to 250 deg. C., he removed the hydrogen, and then by igniting to redness burned the methane. His method in a modified form was at once put to use in this laboratory.

It was difficult to understand at first why a method so simple and seemingly so meritorious should not be in more general use or at least have found a place in textbooks on the subject of gas analysis. The reason is probably twofold. In the first place, Jäger's method of placing the KOH pipette beyond the copper oxide tube may be a source of error as will be shown later, and then the method of standardizing is tedious.

Unsatisfactory Methods for Carbon Monoxide and Hydrogen

In gas anaylsis, the absorption of carbon dioxide with KOH, and thereby the determination of its percentage, is a simple matter. The same is true with reference to the determination of the illuminants (unsaturated compounds) with bromine water, and oxygen with stick phosphorus. However, the absorption of CO with cuprous chloride, and hydrogen with palladium tube is sometimes less successful. At least the results obtained in the absorption of hydrogen with the palladium tube were never very satisfactory in the work in this laboratory extending over quite a number of analyses of a series of coal gas samples. Burning both the hydrogen and methane together was resorted to for awhile; the loss in volume being hydrogen which when burned condenses to water. But this method is open to the objection that there is no check on the carbon monoxide absorption with cuprous chloride; for, the CO present, if there is any, is burned to carbon dioxide and is absorbed and measured along with that due to methane. Furthermore, if there is any ethane present, which is possible, it increases the residue in volume when burned and thus decreases the apparent percentage of hydrogen. However, with Jäger's method, the hydrogen is first removed. Then when the residue from the hydrogen is burned, the increase if any in volume is due to ethane. In this way the ethane can readily be determined.

Of course no distinction is made in this case between ethane and any higher member of the series, which if present would be measured as ethane.

In using Jäger's method, a pipette containing water slightly acid with sulphuric acid, was always used instead of KOH as he directs. It was soon observed that the residue left after burning out the hydrogen at 250 deg. C., frequently contained a small quantity of carbon dioxide. At first it was thought that this small quantity of CO_2 was due to a partial combustion of methane, caused perhaps by its reacting with a small quantity of copper oxide dust that might be present, a very high temperature perhaps not being necessary to bring this about. However, prolonged heating at 250 deg. C., and repeatedly passing the gas did not cause the formation of more carbon dioxide. As a matter of fact, this carbon dioxide when present was due to the burning of the small residue of carbon monoxide which had failed of absorption with the cuprous chloride, either through lack of thorough shaking or too weak a solution.

As a result of this observation, the cuprous chloride pipette has been discarded entirely, and after removing the oxygen with stick phosphorus, the procedure is simply to burn the hydrogen to water and the total of the carbon monoxide to carbon dioxide in one operation at a temperature of 250 deg. The loss in volume is the percentage of hydrogen present. The CO_2 formed is then absorbed by passing into KOH and the loss in volume is the carbon monoxide that was present. The copper oxide is then heated to redness and the residual gas passed. The volume of gas is noted before and after burning. If there is an increase in volume it is due to ethane, and one half this increase is the percentage of ethane present. The total carbon dioxide formed is then absorbed and the residue noted. The loss in volume, accounting for the ethane first, is the percentage of methane present. The residue is the percentage of nitrogen direct.

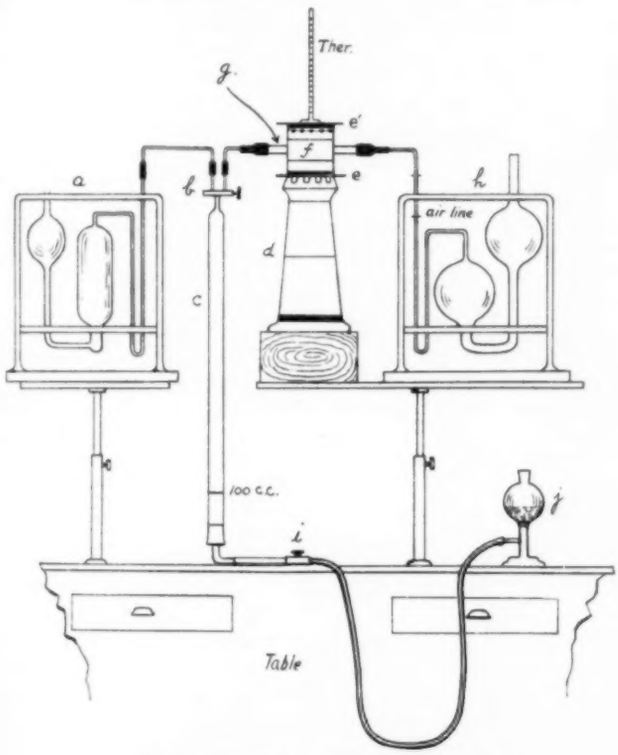


FIG. 1.—APPARATUS FOR GAS ANALYSIS

Description of Apparatus

There follows a description of the apparatus necessary and a detailed statement of the method of using it with the precautions to be taken.

For the method in detail, reference is made to the accompanying illustration. All glass connections are of capillary tubing and the rubber should be thick-walled, tight-fitting. Referring to Fig. 1:

a—Semi-permanently attached KOH pipette (500 gms. to the liter).

b—Two-way stopcock.

c—Gas burette.

d—Craddock burner for convenience. Any burner would do of course.

e—Asbestos board, 1/16 in. in thickness, solid.

e'—Asbestos board, 1/16 in. in thickness, with a hole for a thermometer.

f—A 4-in. asbestos air bath (Eimer & Amend Catalogue No. 2073.

g—Silica tube, 1/4 in. bore, 7 in. long.

h—Pipette filled with water, slightly acid with sulphuric acid.

i—One-way stopcock.

j—Leveling bulb filled with 1% to 2% sulphuric acid.

Method of Procedure

The solution of KOH in *a* having previously been drawn over *nearly* to *b*, the stopcock is closed to *a* and opened to the air. Before the portion of the apparatus shown from *d* to *h* is placed in position, the burette is filled with gas to be analysed until the water sinks about one inch below the 100 cc. mark. The stopcock *b* is then closed and the leveling bulb *j* raised until the gas under pressure has a volume of 100 cc. The stopcock *i* is then closed and *b* opened, whereby the gas is brought under atmospheric pressure and exactly 100 cc. has been taken. The gas is then allowed to pass into *a*, when its carbon dioxide is removed. The water in the burette is now caused to flow out to the air until the entire system including the connection of capillary tubing is filled with water. The bromine pipette containing water saturated with bromine to which a few crystals of KBr have been added is then attached on the right. The gas, returned from *a* to *c*, and back and forth a few times if necessary, is measured by bringing the liquid in the leveling bulb *j* to a level with the water in the burette. Stopcock *i* is closed and the burette is read. The loss in volume is the percentage of carbon dioxide present.

The gas is then passed back and forth from *c* to the bromine pipette until bromine fumes are distinctly visible in the gas on returning to *c*. It is then passed into *a* to remove these fumes and upon being brought back to *c* is measured. The loss in volume is the percentage of "illuminants" (unsaturated carbon compounds) that were present.

The phosphorus bulb is then substituted for the bromine bulb and the oxygen determined in a similar manner; that is, by loss in volume.

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Precautionary Measures

The apparatus as shown in the illustration is then placed in position. Care must be taken to have the right hand opening of the two-way stopcock *b*, as well as the capillary tubing leading from it, entirely free of KOH solution. In fact, *not one drop of KOH should ever be allowed to enter the burette* if it is possible to avoid it, for the reason that one drop can absorb many times its volume of CO_2, causing an error in the hydrogen determination. If any KOH has entered, the gas should be passed into *a* after the oxygen is determined, and the passage way to the air thoroughly washed with the acid solution in *c* and *j* until all KOH is removed or neutralised.

The apparatus having been placed in position as shown, the air line on the capillary tube of *h* is noted. With a little care it can always be brought to the same place. The burner is then lighted and turned down until the bottom of the board *e* glows in one spot in the center from 1 to 2 inches in width. This will generally be found sufficient to bring the temperature of the air in the bath to 250 deg. C. which is the required temperature; from 250 to 260 deg. is sufficient. The silica tube in the air bath is filled in its central portion, about 4 in., with copper oxide, "wire form," crushed until it will pass a 20-mesh screen. The dust formed in crushing should be screened out and none passing 80-mesh should be placed in the tube since it clogs the passage of the gas. The remainder of the tube is filled with an asbestos fiber plug, about one-half inch on each side of the oxide, and then glass wool to each end of the tube. The glass wool is not necessary, except to make the air content smaller, and to help retard the passage of the water if it should by accident be allowed to enter the silica tube on either side. Care should be taken *not to pack* the copper oxide or it will expand on heating and crack the tubing. On the other hand it should fill the tubing entirely if possible, to prevent an empty passage along the top of the tube, and so permit the gas to pass without coming into intimate contact with the copper oxide. A simple way to accomplish this is to place one plug of asbestos in position, then pour in the oxide and tap it gently with the finger to settle it. Then the other plug can be gently pushed into place. The proper place can be seen by holding the tube to the light.

Advantage of Silica Tube

In the former article alluded to above, a hard glass tube was recommended. It was soon discarded in this laboratory for the silica tube which does not bend on heating repeatedly, being more refractory; and, what is of more importance, it does not break if water is allowed to enter it from either side or is condensed from the water formed in combustion. The rubber tube connections to the capillary elbows at either end are of thick-wall "antimony" tubing. No difficulty is experienced from this tubing being burned, as it is protected by the asbestos air bath, through the walls of which the silica tube passes by means of two holes on opposite sides, just large enough to admit the tube and so placed that the center of the tube is about the center of the bath.

The temperature of the air bath having risen to from 250 to 260 deg. C. as shown by the thermometer, the leveling bulb *j* is placed on a shelf above the level of *f* and the cock *i* opened slightly to allow the gas to pass *slowly* through *g*. It must be passed back and forth until it no longer loses in volume. This means that it will have to pass through the silica tube six to eight times. Then, the light being extinguished, and the thermometer and *e*[1] being removed, the apparatus soon cools to the temperature of the room. Then the gas is returned entirely to *c* until the water rises in the capillary tube of *h* to the *former air line*. The cock is then closed and the loss in volume noted. This loss in volume is the hydrogen present in the gas plus the oxygen of the air in the silica tube. All this oxygen has been used up of course. This volume of oxygen has to be subtracted from the loss in volume due to the hydrogen burning, and the difference is the percentage of hydrogen in the gas. (A simple way of determining the air volume of the silica tube or rather of its oxygen content will be given further on.) The residual gas is then passed into bulb *a* and the carbon dioxide formed by the combustion of the CO while heating to 250 deg. is then measured directly by the loss in volume due to the absorption of the carbon dioxide formed. To accomplish this thoroughly, the residue in *c* after being deprived of its CO_2 content is again passed through the silica tube, which of course contains CO_2, and then the whole is again passed into *a*. By repeating this fractional removal three or four times, the CO_2 is reduced to an amount too small to be measured.

The reaction will be found to be complete at the temperature mentioned, provided the gas has been passed a sufficient number of times to burn the hydrogen completely. This can easily be determined as noted above, by passing it back and forth until it no longer loses in volume.

The asbestos board *e* is now removed, the burner again lighted and the tube heated to redness. It may be found necessary to place a single sheet of iron-wire gauze (the sort usually placed under beakers) on top of the bath to heat the tube sufficiently. But this is to be avoided if possible because it may cause heat enough to scorch the rubber connections.

The gas should be passed back and forth about the same number of times as was found necessary in the case of the hydrogen, but in this case can be passed much faster. The light is extinguished, the apparatus allowed to cool and the volume noted. It should be the same as before burning, or at least no less. If less, it shows that the hydrogen and possibly the CO were not completely burned, or that KOH was allowed to enter the burette and up into the capillary tube leading to the silica tube. Any increase in volume is due to ethane or a higher member of the series. This increase in volume, if due to ethane alone, may be read at once as the percentage of ethane. The gas is again passed into *a*, the CO_2 removed and the residue measured. The loss in volume is the percentage of

methane if there was no ethane. If ethane was present, then to determine the amount of methane, the percentage of ethane is doubled and subtracted from the total amount of carbon dioxide formed. For example suppose a mixture of the following gases in the residue: 8cc. of nitrogen, 20cc. of methane and 7cc. ethane, with a total volume of 35cc. Upon burning, the nitrogen, of course, is unaffected; the methane burns to CO_2 but has the same volume as before, but ethane doubles its volume on burning. The total volume will be 42cc.: 42-35 gives 7, the percentage of methane. When passed into KOH, the loss in volume will be 34cc.: 34 less 14 (double the percentage of ethane) gives 20, the percentage of methane. In removing the carbon dioxide, the same fractional method has to be resorted to as in the case of the carbon monoxide determination mentioned above. The sum of the various constituents determined thus far, is subtracted from 100 and the residue is nitrogen; or, the residue after removing the last of the carbon dioxide is the nitrogen percentage when the oxygen content of the silica tube has been added to it.

Correction for Oxygen in Air in Silica Tube

To "standardize" the silica tube and all that portion of the apparatus between *b* and the air line in *h*, the following procedure will be found convenient: Take any gas, such as average city water-gas or coal-gas, known to contain methane to the extent of say 5%, but no ethane, (no sample of gas has been analyzed in this laboratory in the past year that contained more than a "trace" of ethane) and remove the carbon monoxide and hydrogen by prolonged heating at 250 deg. as outlined above. The residue of methane and nitrogen is passed into *a*. The bulb *h* is removed, the burner again lighted, and the cuprous oxide and copper are again oxidised to black oxide of copper by repeatedly passing air in and out. The apparatus is then allowed to cool to the temperature of the room again, and air is once more passed in and out to be sure that the silica tube contains air only. The bulb *h* is then attached as before and the residue of methane and nitrogen burned as outlined above, by passing the gas back and forth two to four times. It is then allowed to cool, returned to the burette and the loss in volume carefully noted. This loss in volume is that due to the oxygen of the air content of the silica tube and accessories in place. This then is the amount to be deducted from the hydrogen percentage and added to the residual nitrogen to determine the percentage of nitrogen. The copper oxide in the silica tube can be reoxidised and used over and over again by passing the air as just described. After it has been used many times, however, it becomes more or less solid on the bottom. It then has to be punched out or dissolved out with acid and renewed. Incomplete combustion of the hydrogen on repeatedly passing it through, marks the stage when renewal is necessary.

Comparative Results

Following are the results obtained in a number of analyses by this method alone, and when combined with the cuprous chloride method.

Gas No. 1.			
	1st sample		2nd sample
CO removed with CuCl	4.6	CO removed with CuO only	14.0
CO removed with CuO	9.6		
	14.2		

Gas No. 2.			
	1st sample		2nd sample
CO removed with CuCl	11.2	CO removed with CuO only	21.9
CO removed with CuO	10.6		
	21.8		

Gas No. 3.			
	1st sample		2nd sample
All CO removed by CuO	16.8	All CO removed by CuO	17.00

Gas No. 4.			
	1st sample		2nd sample
All CO removed by CuO	15.5	All CO removed by CuCl	15.0

Gas No. 5.		
1st sample		2nd sample
Carbon dioxide	0.2	0.2
Illuminants	2.0	2.0
Oxygen	1.4	1.3
Hydrogen	45.1	45.1
Carbon monoxide	19.3	19.4
Methane	24.4	24.6
Residue	7.6	7.4

The results given under No. 5 are those obtained in the course of the analysis, without applying the correction for the oxygen in the silica tube. To do this and obtain the correct results, 1.3 must be taken from the hydrogen in each case and added to the nitrogen.

In conclusion, attention may be called to the fact that in the Jäger method as outlined in the reference, if the CO was not completely removed it would place an error on the hydrogen due to the use of a KOH pipette where bulb *j* is placed in the accompanying illustration. The method of calibrating the silica tube as outlined is simple and accurate.

It may be that the determination of carbon monoxide by this method has already been done by someone else, but if so I have not seen its publication. At any rate, it has not found its way into the various textbooks on gas analysis. After repeatedly experimenting with this method for the past two years, the writer has a decided preference for the method as outlined above over the cuprous chloride, palladium oxide and explosion method, both from the standpoint of accuracy and also of convenience.

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Utilization of Wood Waste—III

STEAM DISTILLATION

By John E. Teeple, Ph.D.

The distillation of wood waste with steam, to recover turpentine and pine oil was at one time quite an industry, and what is more there were certain periods in its history when it was fairly profitable. The rapid development began about 1902 and reached its maximum in 1910 when a very considerable number of plants were in operation. Almost every plant operating during 1910 was able to show a profit for that year and the first few months of 1911. The fall in price of turpentine since then has made the business unprofitable. There is not today a single plant operating by steam distillation alone, excepting where its raw material is sawmill waste, and there are

FIG. 1.—STEAM DISTILLATION PLANT FOR RECOVERY OF TURPENTINE AND PINE OIL

only one or two of these. Since it is an industry that has completed its life course in that form, we can sketch its history with safety.

The process of extracting volatile products from pine wood with steam either saturated or superheated and with either excess pressure or reduced pressure, was patented in 1864 and 1865 by Hull, Leffler, and Emery, and has been repeatedly patented since in its various phases. It apparently did not become a profitable industry, however, until soon after 1900 when turpentine began to stay uniformly above 50 cents per gallon at Savannah.

Then the Krug patent was exploited to a considerable extent in Georgia and Florida. The novel idea of the Krug patent was two outlet pipes, one to remove the turpentine and pine oil, and another to remove the products of destructive distillation. The retort used was the regular horizontal type of hardwood distillation retort such as is commonly used in Pennsylvania

and southern New York. Experience showed that the second outlet for destructive distillation products which was the prime feature of the Krug patent was useless because with this system the plants never proceeded far enough to get destructive distillation products.

Nevertheless, several plants were installed using the general type of hardwood retort with only one outlet. Many improvements in methods of handling the wood, systems of conveyors, systems of discharging, apparatus to empty the retorts, convenient bins and chutes for filling them, enabled some of these plants to continue operating for several years. In this system the wood was brought in usually in 4-ft. lengths, and passed through an edging grinder, commonly known as a hog, to reduce it to fair-sized chips and later in many cases shredders were installed. These, with the engines, retorts, condensers and still house equipment were the largest items of cost.

Another type of retort used was a vertical one holding from

FIG. 2.—LIGHT WOOD IN THE WOODYARD OF A STEAM DISTILLATION PLANT

one to three cords. In some cases the steam entered at the bottom and passed upward through the wood. In other cases it entered at the top and passed downward. Each of these types of retorts variously patented by Gardiner, Atcheson and Snell, were exploited to a considerable extent, especially in Florida. The chief differences were in the direction of passing the steam, modes of agitating and discharging chips, and variation in type of the lower door. All used saturated steam without more than 20 lb. or 25 lb. pressure.

In addition to the horizontal and vertical retorts another variety was set obliquely, consisting of a cylinder holding less than one-half cord set at an angle of about 60 deg., charging at the top and discharging at the bottom. Two plants of this type were installed in Florida.

Another type which was exploited to a considerable extent was the Bethune system, being a type of vertical retort using superheated steam. Some five or six plants were installed from North Carolina to Florida, but the superheaters usually failed on account of insufficient control and were gradually abandoned.

Still another type was the revolving retort suspended on trunnions, the steam entering through one trunnion and the volatile products passing out through the other. This type seemed to offer certain advantages for charging and discharging, and for agitating the product during extracting. The retorts were sometimes cylindrical or frustum shape, and sometimes spherical. This type of retort offers a curious example of the recurrence of ideas and of the failure to profit by experience and other men's work. Some eight or nine years ago when this idea of a revolving retort was first brought to the writer's attention it seemed to offer enough advantages over other methods at that time in use to warrant a fair trial, so two retorts were designed and installed in a plant in Mississippi.

Just about the time these were ready to operate, a patent was granted to a man in Georgia who apparently had the same idea, although as far as I could learn he never progressed far enough to install an apparatus to try out his idea. A few months later the writer was called to inspect a plant in Georgia and found there the same general type of retort in operation. It differed somewhat in shape and construction, but the idea was there as in the others. A few years later the same type of retort was called to his attention at a plant in Louisiana, and within the last three months an inspection of another plant near the Texas border revealed our old friend only recently installed, and ready to operate.

In all these cases the newer plant had apparently been installed without any knowledge of the previous ones, and in each case the previous methods had been practically abandoned after careful study of them before the new ones were built. This is simply one of the many cases that might be cited of a continual repetition of errors resulting in financial loss, and in each case the loss might have been wholly avoided by careful study of what had been done before and what data was available regarding that type of installation.

All these installations mentioned had the retort in the shape of a cylinder or frustum of a cone hung on trunnions. Two installations were also made in Georgia of spherical retorts hung on trunnions. These retorts had a much larger capacity than the cylindrical ones, but the method of operation was the same.

When any of these types of retort was used, the method of extraction removed only the turpentine and pine oil. The chipped refuse when discharged from the retorts was very similar in appearance to the original raw material. A portion of it was used for fuel to run the plant, the remainder being sold either locally or at a distance to railroad shops, tug boats or firms requiring fuel for power. As the wood was very resinous the waste made an excellent fuel when the furnaces were properly equipped to use it.

Steam distillation plants required large quantities of steam for power to run the hogs, shredders and conveyors, and for extraction in the retorts, but their waste materials always furnished them more than sufficient fuel, and from one-half to two-thirds of the total extracted wood was available for sale, provided there was a market.

Many attempts were made to extract further products from this waste in the same retort. In one plant in Florida the retorts were built to stand 80 or 100 lb. pressure, with the idea that steam under this pressure would melt the rosin and allow it to be collected at the bottom of the retort. In another, superheated steam was used on the assumption that the wood would be charred and destructive distillation products obtained, but all of these methods failed in actual practice because of the physical condition of the wood. The very large amount of surface exposed prevented an efficient draining and collection of rosin or tar.

Regardless of the type of retort, all of the steam distillation plants finally came to about the same method of operation. The wood was usually brought in in 4-ft. lengths, as stated, but in one case the hog was large enough to receive full-sized logs, and in some cases stumps were used. The wood cost variously from $1.75 to $4.50 per cord, probably averaging about $2.75.

When the wood was hogged, and sometimes shredded, it was placed in the retorts and subjected to a saturated steam extraction. Experience showed that the best results were obtained by varying the pressure with the amount of volatile products distilling.

At the outset the distillation proceeds normally with little or no pressure. When the volatile products decrease if the pressure is raised 10 or 15 lb. the steam penetrates further into the wood, then when the pressure is decreased the steam and volatile products come out of the chips together and a normal flow of distillate proceeds. This is repeated several times until the extraction is as complete as conditions at that plant warrant.

The maximum pressure usually does not exceed 25 lb. and the time of extraction varies from three hours to six or seven hours, usually not more than four. The crude distillate varies from 10 to 17 gal. per cord, depending on the quality of the wood. Individual cords often give as high as 20 or 25 gal.

but 14 or 15 gal. per cord of crude distillate is considered a very good average unless the wood is especially well selected. The crude distillate separated from the water is then redistilled in stills, with or without fractioning columns, and separated into two marketable products, turpentine and pine oil.

Steam distillation plants deserve the credit for being the first ones to place on the market large quantities of well-refined turpentine and pine oil of a uniform quality and to develop a large market for those products. Steam distillation will be continued probably as a part of other processes, but not as a sole means of utilizing wood, excepting in the case of sawmill waste, which will be discussed in a later article.

It must be said in favor of the steam distillation plants that most of them were built on the very clear understanding that they were not complete processes. The owners in many cases recognized that there was a comparatively small margin in the recovery of turpentine and pine oil alone, even under

FIG. 3.—FEEDING THE HOG

favorable conditions, but the recovery of further products seemed reasonably certain on further investigation.

This recovery did not progress as fast as most of them hoped. Attempts at destructive distillation of the residue revealed many obstacles. Methods of extraction were a long while in being worked out and the ones of these in successful operation will be discussed in another article.

Methods of heating the chipped residue from the outside failed because it was very difficult to make heat penetrate through the mass of sawdust or chips easily, and because when it did penetrate the rosin or tar clung to the chips so closely that it could not be recovered to advantage. Methods of incomplete combustion of the residue by burning a portion of it in order to distil the remainder gave some results in small-scale retorts holding a half cord or more, and this might possibly be worked out. A fair grade of tar and tar oil can be produced in this way.

The steam distillation plants were of varying capacity, from 5 to 10 cords per day up to 80 cords per day, and represented investments of from $10,000 to $200,000. Where the proper location was selected and construction was good, the plants will probably be permanent, after making the necessary additions. Where poor judgment was used in selection of location or where the construction was not substantial, the plants will be very nearly a total loss. Many have already been dismantled or transformed into other types of plants or had additions made to bring up the total production per cord of wood to a profitable basis.

The difficulty of the steam distillation plants as such was a simple one of revenue. Given a cost of wood of, say, $3 per cord and manufacturing cost of $1 to $2, and taxes, insurance, depreciation, interest and overhead charges of, say, another $1, besides the cost of packages and selling, their operation was entirely dependent on the price of turpentine. When turpentine was high so that their products brought from $7 to $11 or $12 per cord, they were happy. If their products bring only $4 they are out of business, and this is their present condition.

Comments on the Report of the Investigation of the United States Patent Office.

By L. H. Baekeland

Whoever is interested in the patent situation of this country should make it his duty to read the unusually interesting publication[1] just issued by the U. S. Government, which covers 624 closely printed pages.

On August 21, 1912, the following joint resolution was passed:

"That the President of the United States be, and he is hereby, requested to cause the accountants and experts from official and private life now or hereafter employed in the inquiry into methods of transacting the public business of the Government in the several executive departments and other executive Government establishments, known as the Commission on Economy and Efficiency, to investigate fully and carefully the administration of the Patent Office with a view of determining whether or not the present methods, personnel, equipment, and building of said office are adequate for the performance of its functions, taking into consideration the present character and volume of business, and also such increase in complexity or volume as may reasonably be expected in the future, and to ascertain and recommend specifically to Congress not later than December 10, 1912, what changes in law, what increases in appropriations, and what additional building accommodations, may be necessary to enable the Patent Office to discharge its functions in a thoroughly efficient and economical manner, and to what extent any expenditures which may be recommended can be met by increases of Patent Office fees.

"All expense incurred in carrying out the purposes of this resolution shall be paid out of any funds in the Treasury of the United States not otherwise appropriated, and the sum necessary for said purposes is hereby appropriated: PROVIDED, That the total expense authorized by this resolution shall not exceed the sum of $10,000."

The report as presented is signed by Frederick A. Cleveland, Walter W. Warwick, and Merrit O. Chance, Commissioners on Economy and Efficiency.

This interesting report enters into the many ramifications of our patent system, shows up its defects and indicates in a practical way possible methods of improvement.

An enormous amount of precise information has been collected by the above-mentioned commission, but what is more impressive is that all this has been accomplished with the expenditure of such a comparatively small sum of money as $10,000, and the report should have been prepared and published three months and a half after the resolution was approved. This in itself is a worthy example of "economy and efficiency."

With these data at hand, in addition with the report of the hearings on the Oldfield Bill (see report No. 1161, to accompany H. R. 23417, by Hon. W. A. Oldfield, House of Representatives, 62nd Congress, 2d Session), any future commission for the revision of our patent system will have its work much simplified.

Certain defects in the handling of the personnel of the patent office are pointed out; for instance, it is mentioned that there are no efficiency records for the examining force and that for many years there have been no examinations for the purpose of determining the efficiency of examiners and regulating their promotions and reductions. Such efficiency records, in conjunction with an examination system, would tend to encourage all employees to do their best work and would make adequate salaries a real and lasting benefit to the service.

Emphatic stress is laid on the unsuitability of the present Patent Office.

"In the preliminary survey preceding the investigation the commission was impressed with the idea that the building accommodations for the Patent Office are so inadequate as to

[1] 62d Congress, 3d Session, House of Representatives, Document No. 1110. Report of the Investigation of the United States Patent Office, made by the President's Commission on Economy and Efficiency, Washington, December 9, 1912.

render extremely difficult any substantial improvement in the work of the office. This impression has been changed as a result of the investigation until it has become a settled conviction that any permanent improvement in the quantity and quality of work done by the office, if done at a reasonable cost, must wait upon provision being made for adequate office accommodations for the force of more than 900 people employed in the Patent Office."

It is a hopeful sign that the Bulkley Bill (H. R., 28,193, 62d Congress, 3d Session) should already have taken up this matter, and intends to provide a $4,000,000 fireproof building. It is questionable, however, whether this sum of money will be sufficient for the purpose. When we take in consideration that the Patent Office has now an accumulated surplus of $7,000,000 over all its expenses, there is no reason why this surplus should not be used for providing a suitable building, and the equipment that goes with it.

Through the courtesy of the authorities of the German Patent Office, I have recently been enabled to inspect that magnificent patent office building in Berlin, and its operation. I have been more than ever impressed with the fact how antiquated and inadequate our Washington Patent Office equipment has become in view of the enormously increased development of our country.

As to the needed changes in our patent laws, the remark is made that the present law is substantially the same as the act of 1836. There is a significant allusion to the conservativeness of lawyers and their opposition against any changes in procedure.

"No change, however much it may be needed to keep pace with changed conditions, can be made without vigorous opposition due to a large extent to that inherent prejudice against disturbing the daily routine life and the inconvenience of adapting one's mind to a new course of action. . . . Recommendations made by the present Commissioner of Patents for the amendment of laws to effect what he considers are improvements in the methods of the office and to expedite its work are opposed by those who do not agree with him as to the wisdom of the changes as well as by those whose financial interests would be affected unfavorably . . . the commission is convinced that any proposed change in the patent law and almost any proposed change in the methods of procedure in the Patent Office will arouse opposition from some persons interested in securing patents."

This statement is very well corroborated by the answers to the list of questions sent out by the Economy Commission to members of the bar, inventors and others interested in matters before the Patent Office. More than 400 replies were received. On most questions there was a considerable divergency of opinion.

But even here it is significant that on question No. 24, relative to the desirability of creating one single Court of Patent Appeals, the answers were practically all affirmative and only a few opposed.

Questions 1, 2, and 3 asking opinions as to the advisability of abolishing one of the two appeals in the office and creating one appeal board of five or six members were answered in the affirmative in about 80 per cent of the replies. The direct negative to these questions was less than 20 per cent. The trend of the opinions expressed lies decidedly toward a single appeal to a board in the office, particular stress being laid on the importance of having the work of the primary examiner performed in a thorough and capable manner. The great majority of answers opposed the sitting of the commissioner, the assistant commissioners, or any administrative officer on this board, the idea being expressed that the commissioner should be left free to devote his time to purely administrative duties.

Answering question No. 14 "to what extent should the decision on such appeal operate as a final adjudication of patentability," a small number answered that the decision should be final, and a few that it should not be final. Some favored it being final only as to adverse decisions. Other suggestions were: It should be final, but not as against newly discovered prior art, in case of litigation under the patent; decision of the Patent Office in interference cases to be final and only to be opened in court upon new evidence found, as the proceedings are far less expensive in the office than in the courts.

Chapter 4 is devoted entirely to an examination of our cumbrous and time-robbing system of interference. The commissioners conclude that it would be better not to make any important changes in our present interference procedure. They seem to overlook the fact that our interference system, as it stands today, hardly protects the "first" inventor if the latter does not possess the abundant cash required to defend his rights against wealthy opponents. Furthermore, a good patent system should be devised in the interest of the nation, the public at large, and not for the limited individual benefit of some inventor or another.

The essence of good patent law is to induce inventors to disclose their invention as promptly as possible, so that the public may have the early benefit of such publication. Our interference proceedings, in conjunction with other defects in our methods of allowing patents, encourage systematic delay in the publication of patents; in reality, a premium is now put on such delays.

All this would be much simpler if in all interference cases priority were allowed to the independent inventor who has filed first his patent, provided he has not copied nor stolen the invention from others.

The German law allows an almost absolute claim to priority to the earlier applicant. This system, therefore, has the great advantage of simplicity over the system in practice in the United States, but it may become a source of injustice in case the latter applicant is really the earlier inventor. However, the latter applicant has only himself to blame if he does not file his application in due time, instead of doing as is now the case, and purposely postpones filing his claims under the immunity of our interference rules. This finally comes down to the question whether it is better to confer the right of priority to the earlier date of application which can clearly and instantly be established, or to the date of conceiving the invention and reducing it to practice, which brings forth all the uncertainties, vagueness, which frequently make our interference proceedings so absurdedly long, complicated and expensive. Our present interference rules are undeniably a real handicap to the inventor of limited means; and our system is detrimental to the best interests of the nation. Indeed, it delays the essential benefit of publication of the invention; furthermore, in some cases it becomes a positive danger to any industrial enterprise when nobody knows whether he is not infringing some unpublished patent rights which have been silently ripening to interference proceedings, protected by all the possible customary delays sanctioned by our rules of practice.

The German and British systems permit to file opposition proceedings within a certain period after a patent is allowed; this is an excellent simplified substitute for our interference proceedings. Members of the German patent office have invariably declared that they attach considerable importance to the maintenance of this system, because it tends to reduce the responsibility of the examiners and to enhance public confidence in the integrity and capability of the board of examiners; this in itself checks criticism to which such sensitive men as inventors are only too prone after they have encountered some disappointments in the office. I have noticed the same uniformly favorable opinion among German inventors, as well as patent attorneys and manufacturers, as to the advantages of the so-called opposition proceedings. Many of them, however, are of the belief that the British system, which allows opposition to a patent during a certain period *after* the latter has been printed and published is, however, more practical and expedient than the German method where opposition has to be filed *before* the patent has been printed, and before copies are readily obtainable except by means of special copies made from the pending application at the patent office.

As another check to worthless patents, the German law provides a safeguard in the method of annulment proceedings.

Such proceedings are quite distinct and independent from any infringement proceedings. Any person, whether sued for infringement or not, may apply to the patent office for the annulment of any patent if he can prove that, at the time of application the subject-matter was publicly known, or was claimed in a prior patent, or was illegally appropriated by the patentee. However, in the latter case, annulment proceedings can be instituted only by the person from whom the invention was appropriated.

In a similar manner, a person who believes he can prove that he is a prior inventor of an invention patented to another, and that the patentee has stolen his invention, can sue the patentee for an assignment of the patent to him.

There is still another safeguard in the German law, called the right of prior use (Vorbenutzungsrecht) which prevents the dangerous possibility which now exists in the United States, that a business enterprise, started in good faith, might be held up at any time on account of a long secretly pending patent application.

In Germany, any person who, at the date of application of a patent, has already used the invention in that country, or made the necessary preparations for such use, has the right to use the invention for the requirements of his own business in his own works; but this right can only be transferred to others together with the good-will of the business.

It should be noted here that sometimes the suggestion has been made to cut down entirely our examination system, as well as our interference system, and to come back to a plain registration system, which was abandoned in this country 76 years ago, but is now known as the French system. The report of the commission says emphatically:

"To return to such a system would be a backward step."

"While there are some persons interested in the subject who believe that the United States should return to the system of registering patents and allow all questions of validity and priority of invention to be determined in the courts, these persons are decidedly in the minority."

In France, a patent may exist for years before an examination is made as to priority and patentability, because such a scrutiny only takes place in case of a patent suit; and even then, the examination is not made by technical experts, but by the ordinary courts, who may or may not decide to hear experts. This absurd system accounts for the scant esteem with which patents are treated in the Latin countries where the French system is in use. The French system has certainly not shown the stimulating effect to invention and enterprise which the American system has introduced in such nations like Germany, England and the Scandinavian countries, which have adopted it in a more or less modified form.

It should be mentioned, furthermore, that in all countries where the French system is used, a patent really begins to be looked upon as serious after the corresponding German patent has been issued. This in itself, is the best tribute to the practical value of the preliminary examination system.

The most striking incongruity in our American system is that after a patent has been granted by a technical board, in infringement cases, this same patent is reexamined as to validity and patentability by a non-technical judge, in a non-technical court. This is the main reason why our system has been defined as "best in the world for lawyers and worst for inventors." (H. W. Leonard, Electrical World, Vol. 51, p. 1055.)

In this relation, the following words are very significant:

"The commission has been unable to see the benefit of the present system by which every question relating to a patent can be litigated in any part of the country.

"When the Patent Office shall have been furnished with adequate quarters and equipment and the most efficient personnel it will be time to consider to what extent the decision of this office ought to be final in patent matters. When that time comes, it may be found advisable to consider whether a patent should not be made valid by law to the extent of giving the patentee a right to an injunction in a case against alleged infringers.

"This would probably require the adoption of the practice of publishing applications when ready for allowance, with the opportunity for anyone to file opposition within a limited time, and if none were filed or it was decided that the patent should issue, it ought to be held valid for all purposes until declared invalid by court. Such a system would probably require also that any person claiming to be injured by the grant of a patent could file annulment proceedings within a limited period. . . .

"There is a general complaint, although the commission is not advised as to the facts of the matter, that a grant of a patent is of no value to an inventor if it is in the interest of any person or corporation of financial strength to infringe it. The results after a patentee wins an infringement suit are often not of real benefit to him, for, if he survives the suit, the actual recovery of adequate damages is exceptional.

"It would seem that the remedy for many of the difficulties of the present situation regarding patents will be found in the development of the Patent Office as an administrative court in which all questions relating to patents, excepting the question of infringement, will be finally decided, subject only to a review by an appeal to a court of patent appeals, with possibly a provision for certain questions going to the Supreme Court. Giving such jurisdiction to the Patent Office, and to one court, will tend to the development of a system of patent law that will be a benefit to the public and a protection to real inventors."

The chapters 5 and 6 are devoted to the classification division and to the scientific library and the search room. They point out the existing defects and contain many excellent suggestions.

The fact that in the files of the patent office, there are still missing copies of 60,000 German patents and of 6000 French patents, is significant. It is true that for a certain period France discontinued publishing its patents.

Several other countries, for instance, Canada and Belgium, do not publish their patents excepting by title. Yet copies of every one of the patents published in those countries ought to be available in our search room; indeed, many of these patents contain important disclosures which have a direct bearing on the validity of the patents issued by the U. S. Patent Office.

It would not involve an exorbitant expense to have typewritten copies made of the patents filed by those countries. A more satisfactory method would be to compel all countries which belong to the International Union to print every patent issued by them. This subject should be taken up at the next meeting of the International Convention, and could hardly cause any opposition, in as far as almost all countries belonging to the convention already print these patents.

A very detailed criticism of the totally inadequate conditions of the building and the equipment of the patent office is made, and specific recommendaions are made for remedying this intolerable condition.

The commission overlooks the fact that the equipment of a new patent office will never be complete unless provision be made for a chemical, a physical and a mechanical laboratory, where simple direct tests can be made under the supervision of two or three experts—a chemist, a physicist, and an engineer; so that many simple questions which now baffle the examiners in their decisions on patentability could easily and quickly be decided by a direct test. Such a laboratory exists in the German Patent Office building.

The report is accompanied by several appendixes, all of which make interesting reading. These appendixes are as follows:

A. History of the United States Patent System.
B. United States laws and rules of practice relating to patents, trademarks and prints and labels.
C. The German Patent law.
D. The English patent law.
E. Discussion of the German patent law and patent procedure.
F. A comparison of the patent laws and procedure in Germany, England and the United States.
G. Methods of examining applications.

H. Publications of the patent office.
I. Statement of the business of the patent office.
J. Bibliography of the United States patent office.
K. Classification of patents and printed publications.

Specially instructive are appendixes E and F, written expressly by Prof. Dr. Albert Osterrieth and Mr. A. du Bois-Reymond, both of Berlin, and leading authorities on the subject. The latter paper not only compares the advantages of the laws of Germany, England and the United States, but points out their relative defects which experience has proved to exist and furthermore discusses the differences in procedure in these countries.

This discussion of the advantages and defects of the leading patent systems of the world, is a most valuable contribution and will be very useful in the consideration of changes that are suggested in the law and procedure of the patent system of the United States.

It is specially interesting to read the comments on the practical value of the so-called compulsory working clause or compulsory license clause in Germany and England, in view of the fact that an attempt is being made by the Oldfield Bill, and other proposed measures, to introduce this innovation in the United States.

In Germany, the compulsory working of patents has practically been abandoned and is merely kept up as a club to compel some countries like England, France and others into reciprocity treaties waiving the compulsory working of patents.

To the credit of the present commissioner of patents, it should be mentioned that he has already negotiated a treaty to that effect with Germany.

The United States, in accord with Germany, might have a decisive influence in abolishing this compulsory working clause in foreign countries, so irksome to American patentees, by passing a law whereby foreigners are compelled to work their patents in the United States, unless their country waives compulsory working of patents issued to American inventors.

The first Oldfield Bill had a clause of the kind, but it was so erroneously worded that in some paradoxical way, it gave superior advantages to foreigners over American inventors.

As to the compulsory working and compulsory license in Germany, we might best quote Prof. Osterrieth:

"In fact, very few patents have been revoked in Germany for non-working (one patent out of more than 2000). On the other hand, the working clause always threatens the patentee, hanging over his head like the sword of Damocles, as a great expert in patent matters once said. Besides, experience has shown that petitions for revocation are mostly entered by patent infringers, using the working clause as a defense.

"This explains why in the last 15 years, there has been a very strong movement in Germany for abolishing the working clause. This first induced the German Government to abolish the obligation of working by international treaties in the relations with Italy, Switzerland, and with the United States. And finally the old working clause of the first patent law was abolished by the law of June 6, 1911.

"Since this law came into force, *i. e.*, July, 1911, no obligation of working a patent exists in Germany. No patent can be revoked on the ground that it is not being worked. Yet, considering the legislation of some countries which threaten foreign patentees with revocation of these patents, if the invention is exclusively or mainly manufactured abroad, the German Government resolved to provide for a similar clause, applicable at least against such countries where German manufacturers have to suffer from such legislation.

"It has therefore been stated in above-mentioned law of 1911, that a patent can be revoked, as far as international treaties do not provide to the contrary, if the invention is exclusively or mainly manufactured or carried out outside the German Empire, and its possessions. This clause does evidently not apply to citizens of the United States, even if an American has acquired a German patent from a patentee belonging to another country. Yet this might lead to abuses. Therefore, there has been provided a complimentary clause, saying that transferring a patent to another person is ineffective if this transfer has been made only with the intention of avoiding revocation.

"According to the former German law a patent could be revoked, after expiration of three years from the date of granting, *if it appeared in the public interest* that permission to use the invention be granted to others, but the patentee refused to give such permission in return for adequate remuneration with adequate security. This clause has proved most ineffective as *no single case of revocation of a patent on this ground has been known.* Therefore, when the abolition of the working clause was discussed, it was suggested to do away also with that clause on the obligation of granting licenses. It was objected that cases might happen where the use of a patented invention by others than the patentee could be necessary *for the public welfare*, for instance, manufacturing certain drugs in case of an epidemic.

"This consideration seemed to justify the insertion of the following clause in that new law of 1911:

"'If the patentee refuses to grant license to another for using the invention upon the offer of an adequate compensation and security, such grant for using the invention can be allowed (compulsory license), if such granting seems necessary in the public interest. The grant may be limited or subject to special conditions.'"

This detailed report of the Commission on Economy and Efficiency is supplemented by some specific recommendations as follows:

1. That a new building specially designed, equipped, and furnished be constructed on a suitable site in the city of Washington, for the exclusive use of the United States Patent Office.

2. That the number of officers and employees of the United States Patent Office be increased, and the increases and readjustments of salaries be made as shown in detail in this report involving an increase of 36 in the number of employees and a total increase of $236,550 in the pay roll.

3. That the Commissioner of Patents be the head of the Patent Office; that his duties be the same as are now prescribed by law, excepting that he be relieved from the consideration of cases on appeal; that he be aided by an assistant commissioner and seven supervising examiners in the administrative work, including control of the methods and procedure of the 43 examining divisions in the allowance and rejection of applications for patents.

4. That one appeal within the United States Patent Office be eliminated; that the number of members of the board of examiners in chief of the Patent Office be increased from three to five; that all appeals within the office be taken to that board; that its decision be the decision of the Patent Office; that the appeal therefrom be to the Court of Appeals of the District of Columbia, as now allowed from the decisions of the Commissioner of Patents.

5. That the fee for filing an application be increased from $15 to $20; that appeal fees be adjusted to the conditions arising from the elimination of one appeal; that a fee of 25 cents be charged for each additional patent, etc., included in one instrument presented for record; that all fees be paid directly to the Patent Office; that refundment of fees paid by mistake be made by the financial clerk and not by warrant from the Treasury.

6. That the life of a patent be so limited as to expire 19 years from the date of filing the application therefor, excluding the time (not exceeding two years) during which an application may be involved in interference.

7. That the work of reclassifying patents and digesting of printed publications, and providing facilities for simplifying and making more accurate the search, be recognized by an appropriation for an adequate force to be employed upon such work.

8. That the subscription price of the Official Gazette be increased from $5.00 to $10.00 and the method of distribution to libraries be changed to reduce the number of copies so distributed.

9. That all the work of producing the publications of the

Patent Office, including copies of patents, be done at the Government Printing Office.

10. That an appropriation be made for the repair of the rooms occupied by the Patent Office and for the installation of suitable lighting and ventilating facilities and for the purpose of new furniture and equipment.

Yonkers, N. Y.

Essentials in the Making of the Finer Steel Sheets

By John G. Homan

Wood is combustible, fragile, increasing in price, decreasing in quality and availability; accordingly other materials are naturally sought for substitutes. Where automobile bodies, certain furniture, interior trimmings of houses, etc., are concerned, sheet steel, with clean, highly finished surfaces has found welcome. The relation this class of sheets bears to the common and familiar "One Pass" material is about the same that hemlock or elm might bear with rosewood or lignum vitae.

For instance, where the architect calls for the splendid walnut-finished doors, etc., for the modern fireproof hotel, he will be satisfied with no less than a product which requires careful inspection to determine the body material. His art calls for finish and imitation, while his engineering demands durability and fireproofing qualities.

The sheet steels for this work have much in common with the woods they supplant; they are necessarily of fine texture or grain, and the practice which produces the commoner material can no more produce satisfactory auto sheets or metal furniture steel than a cupola could make armor plate.

It is the intention of this article to relate something of the essentials in the manufacture of these highly finished steels, as is carried on by a concern devoting all its energies and fabricating equipment to these products.

The steel making is carried on in a plant equipped with basic open-hearth furnaces of moderate capacity; moderate capacity because of the satisfaction obtained in the cleaning action of such furnaces, due likely to the larger value of the ratio of slag surface to bath volume and other considerations not available in the larger furnace of 60 or 100 tons capacity. The fuel is natural gas and has therefore the advantage of easy regulation, sharp thermal value, and absence of sulphur. The raw materials are as usual, pig iron and scrap, both of which are selected and secured from sources which insure certain regularity of composition.

The policy in steel making of producing clean steel from clean materials, rather than making questionable material and answering the question in the ladle, with all sorts of deoxidizers, etc., is followed, as it is essential that the steel be clean; and the cleanliness is to be determind not by the testing machine, but by the material accepted or rejected by the finisher.

The casting is done by the bottom cast method and has many advantages already familiar to those who read this article; the disadvantage is expense. Here, as in all the processes, care is needed, the runner brick and pouring system must be clean. Compressed air is useful in removing particles of brick, etc., which would otherwise contaminate good steel.

When cast and cool the ingots are stripped from their molds and sprues connecting them broken, separating the ingots, which are then piled in stock. From this stock they are taken to the heating furnaces preparatory to reduction. During the heating a rather heavy covering of scale is formed.

The first mechanical reduction is accomplished by the blows of an 8-ton steam hammer. The action of the hammer on this steel is very much the same as for the fine hammered tool steels, it accomplishes a homogenous refining of the grain; a refinement which no rolling method in the writer's experience has been able to show. At the first blow of the hammer almost all of the scale envelope falls away from the ingot, carrying with it any surface impurities from unclean molds. The final blow finishes the 1500-lb. ingot into billets of section about one-fourth that of the ingot. From the hammer these billets are conveyed on buggies by men to the bar mills. The billets are not reheated after leaving the hammer.

At the bar mills the billet is roughed down in five passes to a suitable size for the finishing stand of rolls, in which it is finished in five more passes to a bar 8 in. wide and of weight corresponding to the sheets to be made from it. At all times when possible water is used on the hot bars; one of its important effects being the tendency of the steam formed to remove surface scale, if any. The long, finished hot bars are conveyed one at a time to a bar shear where sheet bars are cut to length under the inspection of a man who throws aside any bad looking material. The selected bars are allowed to fall into a vat of circulating cold water, where they are further cleaned by the explosive action of the steam formed on their surfaces.

For certain classes of highly finished product the bars thus made are cleaned by pickling in a solution of sulphuric acid, but in the manufacture I am outlining the steel is *made clean* and every effort is used to keep it clean, so that ordinarily little is gained by the bar pickling.

The bars are rolled into sheets by very heavy mills; the main advantage being the ability of such rolls to produce sheets from bars relatively cooler than the usual practice would permit. The rolls in question are 38 in. in diameter and 44 in. long and weigh approximately 12,000 lb. each. The stands are correspondingly heavy and designed to allow as little elasticity as possible. Brushes and water are used freely at these mills.

The large roll has the advantage of the larger periphery and the consequent longer life of a turning, and as they are turned up much more frequently than the ordinary sheet mill rolls, they will produce material with much more uniform gauge because of this fact.

While in ordinary practice the roughing or "soft" mills are little regarded as to condition, in this procedure utmost care is bestowed on them. Attempt is made to produce a sheet so uniform in physical and mechanical make-up at the hot mills that the subsequent treatment will have a uniform action because of a uniform base.

The furnace in which the bars are heated, as well as that in which the sheet packs are heated, requires care that appeals to the tonnage producer in the ordinary mill as impractical. Here care is required that can be exercised by a skill that comes only from experience. The design of the furnace, the kind of fuel and the quality of flame are very important considerations.

The finished packs are slowly cooled and no water is used on them as is sometimes required to assist the opening process. When cool the packs are sheared and the sheets separated or "opened." The temperature of rolling, together with the chemical composition of the sheets allow easy "opening."

After shearing, various methods or procedures may be followed, depending upon the requirements of the user. It is in order to say that these requirements are very different even with manufacturers using the materials for the same purposes, so that a finished sheet which appeals to the idiosyncrasies of one purchaser may be too hard, too soft, or lack certain surface characteristics that may be desired by his fellow craftsman.

After shearing, cold rolling is usually in order; if so, the material is conveyed to the cold rolls. In the works where fine goods are made two classes of cold mills are required, one for the beginning or rough work and the other for the finishing passes. Both kinds require great care because the little waves and small marks that are always present in the very best grades of common sheets are absolutely out of question here. When a sheet is fed into the cold rolls the sharp edge of the end of it marks the rolls so that after a little while the rolls become almost covered with parallel lines from this cause, so that the following sheets have these marks transferred to their surfaces and are objectionable. The marks must be removed,

and it is the function of the finishing cold mills, and intermediate processes, to remove them as well as many other errors that emphasize themselves only when one has to deal with the rigid requirements of the usual buyers of highly finished sheets.

After cold rolling one or more passes through the cold mills, the sheets may be cleaned by pickling in acid, washing and drying. This is essentially a hand work and requires all the patience and skill of a good launderer because some sheets respond quicker than others and some require attention that machines of any type familiar to the writer cannot give.

After this important operation, for a certain finish, more cold rolling may be in order, or for another requirement annealing may be required. This latter operation is carried out by placing the sheets in a pile on a cast steel plate, or "bottom," and covering the pile with an air-tight wrought-steel box. This combination is sealed at the junction of the box and bottom with sand and run into a furnace where heat is applied to the extent required. This operation is usually in the hands of an experienced foreman who knows how to apply the heat properly, which means not only time, temperature and uniformity, but the expert knowledge of the phenomenon that what appears from outward signs to be going on inside of the annealing box may or may not be going on. The annealer must know when and how to draw the boxes and he must, of course, be familiar with the causes that produce even the little effects usually overlooked entirely by the ordinary sheet annealer. Lately we have assisted the skill of this workman with that very useful tool of the metallurgist, the pyrometer. Equipment is here, as usual in this manufacture, of utmost importance; fuel, furnace design, type of boxes, etc., require detailed study.

The operations mentioned above in brief detail constitute the more important ones, and for the different kinds of goods they are applied in different combinations of routine. The thing least altered is chemical composition of the steel, probably because the one used is known to give the better surface tendencies and likely also because, for the use intended, sufficient hardness or softness can be had by skillful working.

Besides the above-mentioned processes there is an adjunct one connected with that of annealing and which is known as "deoxidizing." While the theory of this process is apparently simple, the real work is more or less difficult. The process is a matter of breaking up by heat the oxide film which covers the usual sheet of steel and preventing its reformation while cooling by filling the box in which the heating is done with a reducing or neutral atmosphere, usually natural gas. The process should leave a very clean, silvery surface, and it is sometimes spoken of as "white annealing."

Then there are operations for final levelling of the sheets, of these the "patent" or "stretcher" leveller is most effective, though very wasteful. The patent leveller is a machine which grips the ends of a pack of three or four sheets in vice-like jaws and stretches them flat by means of a hydraulic system. This operation hardens the material somewhat. Where the jaws of the machine grip the sheets indentations are obviously made which require another shearing operation as well as the loss of the marred ends of the sheets.

The roller leveller is a small machine consisting of two series of parallel rolls about 5 in. in diameter. The rolls are so placed that when a sheet is passed through the machine one series of rolls is above the sheet and one series below. The purpose of this process is to remove buckles, bellies, etc., from sheets with minimum hardening effect.

The composition of steel which seems to offer the best all around usefulness when subjected to the processes above mentioned seems to be one of the more pure iron content, *i. e.*, carbon about 0.12 per cent and manganese about 0.40 per cent, with sulphur in the neighborhood or perhaps below 0.025 per cent.

For auto sheets of the highest class considerable cold rolling is necessary, the first two or three passes reduce the thickness somewhat and seems to definitely refine the grain. The hardness produced can be removed by annealing, which process is up to the skill of the annealer, as the degree of hardness in this material is subject to the peculiar wants of the user.

To prevent corrosion in transit, etc., these finished materials are often oiled with an acid-free heavy oil.

Follansbee Brothers Company.
Pittsburgh, Pa.

Laboratory Apparatus for the Exact Analysis of Flue Gas

In many manufacturing plants apparatus are now used and are proving very successful for determining a few of the constituents of flue gas. In some automatic CO_2 recorders are employed, in others samples are collected at intervals or during extended periods of time and analyzed by means of various types of gas-analytic apparatus, such as those devised by Orsat, Hempel, Elliot, and Bunte. With any of these apparatus results can be obtained by which the performance of a boiler furnace can be intelligently supervised and the introduction of these apparatus has resulted in important advance in efficiency of operation.

If for technical research the exact composition of the waste gas is desired, however, the analysis is more complicated and has been the subject of an investigation at the Bureau of Mines. The results of this investigation are described in a paper by **George A. Burrell** and **Frank M. Seibert** (Technical Paper 31 Bureau of Mines). From this paper the following extracts are taken:

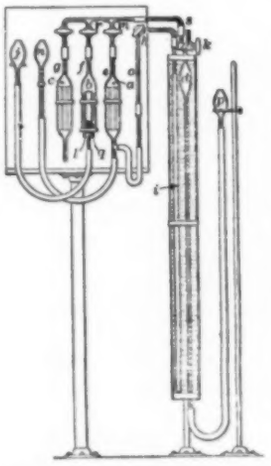


FIG. 1.—LABORATORY FORM OF APPARATUS FOR THE EXACT ANALYSIS OF FLUE GAS

For the determination of carbon dioxide, oxygen, carbon monoxide, hydrogen and methane in flue gas during combustion tests conducted by the Bureau of Mines the authors have used the apparatus shown in Fig. 1.

This apparatus is a modification of the well-known Haldane apparatus. (J. S. Haldane and C. L. Foster, The Investigation of Mine Air, 1905, p. 100.) It has been simplified from the original pattern in that the pipettes are all placed in a common train. Three 3-way stopcocks have been replaced by three 2-way stopcocks, and a simpler pipette has been provided for the alkaline pyrogallate solution. The essential features and the method of working remain the same as proposed by Dr. Haldane for analyzing mine air.

The burette *d* conrains mercury, and has a total capacity of 21 c.c. The bulb of the burette, which is not graduated, has a capacity of 15 c.c. The stem of the burette has a capacity of 6 c.c. and is graduated at 0.01 c.c.

The three pipettes shown are the potassium-hydroxide, *a*; the slow-combustion, *b* and the alkaline pyrogallate, *c*. The reservoir of the alkaline pyrogallate pipette is placed back of the stand and is provided with a rubber bag to prevent access of air to the alkaline pyrogallate solution.

The surfaces of the liquids in the reservoir bulbs must, in all cases, be at the same level as the surfaces of the liquids in the corresponding pipettes when the liquids are brought to the marks represented at *e*, *f*, and *g*, in the figure. This condition is necessary to prevent a difference in pressure acting against the columns of the liquids in the pipettes when the gas sample is transferred from the burette to the different pipettes.

To perform an analysis, the liquids in the three pipettes are brought exactly to the marks, *e*, *f*, *g*, on the capillary tubes. If an analysis has been recently made, this preliminary step is unnecessary, but if the apparatus has stood unused for some time, the solutions usually have to be adjusted before making an analysis. The three-way stopcock *H* is turned to communicate between the outside air and the compensating tube *i*, so that the pressure in the tube may equal that of the atmosphere, and

is then turned to communicate between the tube *i* and the column of liquid in the capillary tube *o*. The gas sample is transferred to the burette by displacing it with mercury.

The first 5 or 10 c.c. of gas drawn into the burette *d* from the sample container is not retained, but is used to sweep the air out of the connections between the burette and the sample container. For this purpose an extra 3-way stopcock is placed between the burette and the sample container. Gas is then drawn into the burette until the mercury is depressed somewhat below the 21 c.c. mark; then, with the burette stopcock *k* closed, the gas is placed under pressure by slightly raising the level bulb *p*. Next the burette stopcock *k* is opened to the air for a second to bring the gas in the burette to atmospheric pressure. Before reading the volume of gas inclosed, the stopcocks at *k* and *m* are turned so that communication is made between the burette and the potassium-hydroxide solution pipette *a*.

If the directions for manipulating the apparatus have been followed with care there is only a slight movement of the potassium-hydroxide solution at *e*. If, through failure to follow directions exactly, there is a decided movement of the solution, error will ensue. A resulting increase of pressure causes an absorption of some carbon dioxide before the reading is taken, and a decrease of pressure pulls the potassium-hydroxide solution into the horizontal capillary tubing, thereby ruining the value of the combustion data subsequently obtained.

By slightly raising or lowering the level bulb *n* the surface of the potassium-hydroxide solution is brought exactly to the mark *o* in the capillary tube.

The potassium-hydroxide solution is brought to the mark *e* above the pipette by a slight movement of the mercury in the burette, and the volume of gas is read to the third decimal place. The burette can be read to 0.002 c.c. The gas is then passed into the potassium-hydroxide pipette, and the carbon dioxide is entirely removed by passing the gas back and forth between the burette and the potassium-hydroxide pipette about four times. In no case should the mercury in the burette be raised above the stopcock *k*. The reduction in volume of the sample caused by the removal of the carbon dioxide is recorded.

The sample is next passed into the slow-combustion pipette *b*. The platinum coil in this pipette is No. 30 (B. & S.) wire and is in contact with platinum wires in two glass tubes filled with mercury. Each wire is sealed through the bottom of its tube and to the projecting ends are attached the copper wires for carrying the electric current. The platinum coil is brought to a white heat by a current of about 5 amp. About two minutes is required for the complete oxidation of the combustible gases contained in the sample. The gas remaining in the capillary tubing is brought in contact with the platinum spiral by passing the sample back and forth between the burette and the combustion pipette several times, care being taken that the mercury in the combustion pipette is not raised to the platinum spiral in the pipette. Finally, the current is broken and the pipette allowed to cool. Cooling is hastened by playing a stream of compressed air upon the pipette. When cool, the gas is transferred to the burette and the contraction in volume is recorded.

The residual gas is then passed into the potassium-hydroxide pipette *a* to absorb the carbon dioxide produced by the combustion. After the contraction in volume caused by this absorption has been measured, the gas is passed into the pipette *c* containing the alkaline pyrogallate solution to absorb oxygen.

At this stage of the analysis the capillary tubes are manifestly filled with residual gas that contains oxygen. In order to avoid sweeping them out by dilution and to obtain the last trace of oxygen, the mercury in the combustion pipette is brought to the stopcock *r* by raising the bulb *j* and the potassium-hydroxide solution is brought to the stopcock *m* by raising the bulb *n*. The oxygen is completely absorbed by passing the gas back and forth between the burette and the alkaline pyrogallate solution five or six times.

The alkaline pyrogallate solution is finally brought exactly to the mark *g*; the mercury and the potassium-hydroxide solution are brought to their respective marks at *f* and *e*, and the reduction in volume of the gas is measured. The oxygen percentage thus determined represents the oxygen remaining after a certain volume has been consumed in the burning of the combustible constituents. To determine the volume of oxygen consumed, a separate determination is made by drawing a fresh portion of the sample of gas into the burette and passing it in turn into the potassium-hydroxide and alkaline pyrogallate solutions and measuring the reduction in volume produced. The total oxygen content thus determined, minus the residual oxygen already found, represents the oxygen consumed.

Other burette readings are made in a manner similar to that in which the original volume of the sample was read. After combustion, for instance, the gas is withdrawn to the burette and the mercury in the combustion pipette brought exactly to the scratch at *f*. The stopcock at *r* is closed and the one at *m* opened. The solution in the capillary tube is brought to the scratch *o* by raising or lowering the level bulb *n* and the solution in the potassium-hydroxide pipette *a* is brought to the mark *e* by a slight movement of the mercury in the burette. The burette is then read.

From the contraction, the carbon dioxide produced by the combustion, and the oxygen consumed, the methane, carbon monoxide, and hydrogen may be calculated by the following well-known equations:

$$CH_4 + 2O_2 = CO_2 + 2H_2O$$
$$2H_2 + O_2 = 2H_2O$$
$$2CO + O_2 = 2CO_2.$$

An analysis such as is described above can be performed in 35 minutes.

Minor Details.—Heavy-wall pure-gum tubing is used on all connections, and the stopcocks are kept well lubricated. The stopcock grease used is made by adding one part paraffin and three parts vaseline to one part pure gum rubber. The rubber is melted, the other ingredients are added, and the mixture is then heated to melting for several minutes. The inner walls of the burette and the compensating tube are kept slightly moistened with water. The instrument should be protected from drafs and the water in the water jacket, each time before burette readings are made, is agitated by blowing air through it, thus insuring uniformity of temperature throughout the water jacket.

It is desirable, if compressed air can be obtained, to keep air bubbling through the water jacket during the entire analysis.

TABULATED RESULTS OF DUPLICATE ANALYSES OF A SAMPLE OF FLUE GAS

Observation or Calculation	First Analysis				Second Analysis			
	Volume 1	% 2	Volume 3	% 4	Volume 5	% 6	Volume 7	% 8
	c.c.		c.c.		c.c.		c.c.	
Volume of sample taken	20.178	..	20.300	..	20.379	..	20.300	..
Carbon dioxide (after caustic polish)	18.181	9.90	18.291	9.90	18.361	9.91	18.288	9.91
Oxygen (after pyrogallate solution)	....	..	16.373	9.45	....	..	16.373	9.43
Total volume of residual gas	18.181	..	....	..	18.361	..	....	..
Volume after burning	18.006	..	....	..	18.186	..	....	..
Contraction	0.175	..	....	..	0.175	..	....	..
Volume after carbon-dioxide absorption	17.800	..	....	..	17.982	..	....	..
Carbon dioxide produced	0.206	..	....	..	0.204	..	....	..
Volume after oxygen absorption	16.022	..	....	..	17.191	..	....	..
Residual oxygen	1.778	..	....	..	1.791	..	....	..
Oxygen consumed	0.128	..	....	..	0.127	..	....	..
Methane	....	0.01	....	..	....	0.02	....	..
Carbon monoxide	....	1.02	....	..	....	0.98	....	..
Hydrogen	....	0.24	....	..	....	0.23	....	..

Bubbles of water mercury must be kept out of the horizontal capillary tubing. It is scarcely necessary to add that a trace of potassium-hydroxide solution left in the tubing will ruin the

value of the combustion analysis. The tubing from *o* over to the compensating tube *i* must also be kept clear of detached bubbles of the potassium-hydroxide solution.

Recording of Results.—The following results of two duplicate analyses show the details observed and the calculations made. As will be observed, the analyses are of flue gas from a boiler furnace operating under very poor conditions:

In column 1 are shown data representing the changes in volume of the gas sample at different stages of the analysis.

In column 2 are shown the proportions of carbon dioxide, carbon, monoxide, hydrogen, and methane calculated to a percentage basis.

In column 3 are shown the observed data involved in the separate oxygen determination.

The final figures representing the percentages of carbon dioxide and of oxygen are shown in column 4.

In columns 5, 6, 7, and 8 are shown the readings and results of the duplicate analysis.

From the combustion data calculations were made to show the errors that might result if the oxygen consumed were not determined, and the results were calculated to carbon monoxide and methane. The possible errors are indicated below:

POSSIBLE ERRORS FROM FAILURE TO DETERMINE OXYGEN CONSUMED

	First Analysis, Per Cent	Second Analysis Per Cent
Carbon monoxide	0.78	0.76
Methane	0.24	0.24

A New House Organ

We have received No. 1 of Vol. 1 of *The Abrasive Age*, "devoted to the better use of abrasives," and published monthly by the Carborundum Company, of Niagara Falls, N. Y. It is interestingly written, well printed, and nicely illustrated. There are articles by F. B. Jacobs on disc grinding, by A. B. Maine on garnet paper and cloth in the chair factory, by Edward Edwards on glass grinding, and numerous smaller notes. We wish this new journal a happy and useful career in furthering the already great popularity of artificial electric-furnace abrasives.

The selection of proper screens in any industry where products are sized is one of the most important branches of that industry. This subject has received close attention at the hands of W. S. Tyler Company, Cleveland, Ohio, and is well treated in that company's artistic catalog No. 40. The company is scientifically equipped to render screen users all possible assistance in solving their screen problems. A testing laboratory contains 2000 variations of screen sections, different in size of opening, mesh, metal, and diameter of wire. From the catalog it should be possible to select a screen suitable for almost any purpose, as over 200 styles are illustrated and fully described. The catalog contains also some valuable information and data on screen analysis, with methods of plotting such an analysis in two different forms. Special attention is given to standard testing sieves, with openings increasing in the ratio of the square root of 2, or 1.414. For closer sizing another series has openings increasing in the ratio of the fourth root of 2, or 1.189.

The production of copper in Russia has increased greatly in the last few years, according to a report from the Consul-General at Moscow. Seven years ago the output was about 10,000 tons, and did not meet half the requirements of the country. In 1907 the production was 14,750 tons; in 1910, 23,000 tons; 1911, 26,500 tons, and the estimated output for 1912, 30,000 tons. The production of electrolytic copper in Russia started in 1907, and in 1911 another plant was started. At present all domestic requirements of electrolytic copper are met with metal of Russian origin.

The Determination of Sulphur in Ferro-Vanadium

By Wm. W. Clark

Sulphur cannot be determined in ferro-vanadium by the evolution method, as varying amounts of the sulphur remain in the evolution flask as insoluble sulphides. Every possible method of evolution has been tried and it has been found, that in each case, some of the sulphur remains in the flask. The modification which gave the best results was the addition of 1 gram metallic aluminum and 10 cc. hydrofluoric acid to the alloy in the flask, using dilute hydrochloric acid as the dissolving agent. About 80 per cent of the total sulphur was the maximum that could be evolved, while on some grades of alloy the results would run as low as 10 per cent of the total sulphur. C. M. Johnston mentions this fact in his book on the Chemical Analysis of Special Steels, Steelmaking Alloys and Graphites, page 16.

The barium chromate method, which is used to determine sulphur in coal and coke, was tried, but it requires, first, the removal of the vanadium, and finally the iron. The method necessitates several filtrations and unless the conditions are kept absolutely constant, the results vary beyond checking limits.

The old gravimetric method is the only one by which the sulphur can be accurately determined in ferro-vanadium. The details of the method in use are as follows:

A factor weight (1.373 grams) of the finely powdered alloy is dissolved in a mixture of 40 cc. concentrated hydrochloric and 10 cc. concentrated nitric acid, and 1 gram sulphur free sodium chloride. The solution is evaporated to dryness, moistened with strong hydrochloric acid and again dessicated. If a red granular precipitate settles out during evaporation, causing the solution to spit, more hydrochloric acid must be added. If the alloy is refractory to these acids, it is fused in a nickel crucible with a mixture of 3 grams magnesium oxide and 7 grams sodium peroxide. The magnesium oxide is used to deaden the reaction, as powdered ferro-vanadium, when heated with sodium peroxide, combines so rapidly with the oxygen from the peroxide, that it melts the crucible and is almost an explosion. The fused mass is dissolved in the least possible amount of water, acidified with strong hydrochloric acid, adding about 40 cc. excess, and evaporated to dryness.

The residue in each method is dissolved in 60 cc. strong hydrochloric acid, diluted to 400 cc. with boiling water, and the silica removed by filtration, washing the precipitate free from salts with hot 3 per cent hydrochloric acid. The filtrate is heated to boiling, 10 cc. of a saturated solution of barium chloride added, the solution boiled for 10 minutes, and set aside over night in a warm place. It is then filtered on a OO Munktell filter, washed free from salts with a hot 1 per cent hydrochloric acid, 1 per cent barium chloride solution, and finally with a hot 3 per cent hydrochloric acid solution until free from all soluble barium salts. When washing the barium sulphate precipitate from the soda fusion, it must be remembered that soda is hard to remove from barium sulphate, and the washing must be continued with the barium chloride solution considerably longer than when washing the precipitate from the acid solution of the alloy.

The precipitate and paper are ignited in a weighed platinum crucible cooled and weighed. The difference in weight, multiplied by 10, gives the percentage of sulphur in the alloy.

The above method, while slow, does not require much of the operator's time, not taking more than 3 hours of actual work on 6 determinations. It is very accurate, and results should check within 0.005 per cent sulphur.

American Vanadium Co., Bridgeville, Pa.

Congress has authorized the expenditure of $2,596,000 for a building to accommodate the Geological Survey, the Bureau of Mines, the Reclamation Service and the Indian Office, all bureaus of the Interior Department. Plans for construction will go forward at once.

The Heat Balance of the Open Hearth

An Account of an Elaborate Heat-Efficiency Test Made on Two 60-ton Open Hearth Furnaces. By Sidney Cornell.

There is a deplorable lack of any printed data in English in relation to the open-hearth furnace from a metallurgical standpoint. Hence, the following is published with the hope that the data given may be criticized, and lead to further information on this most important subject.

The modern blast furnace, as operated today, is run with due consideration for known chemical and metallurgical laws, but this is not so much the case with the open-hearth furnace. The advantage of a change in this direction can be appreciated, and the time has come for betterment of the conditions as existing in open-hearth practice from a metallurgical standpoint.

The data from which this test was figured was obtained at the Duquesne Works of the Carnegie Steel Company. The open-hearth furnaces tested were two 60-ton furnaces, together with their four Hughes mechanically operated gas producers. It was necessary to test two furnaces due to the fact that the gas flues from the four producers are not separated, hence the gases from four producers feed two open-hearth furnaces. The hearths of the furnaces tested were 32 ft. 0 in. x 14 ft. 9 in., and the Hughes producers were of the standard design for that type, and capable of gasifying 25 lb. of coal per square foot of grate area per hour.

The test covered an entire week, starting when the steam was turned on the producers, the Saturday of one week, and terminating when the steam was turned off the producers the Saturday of the following week, thus covering a period of 161 hours and 32 minutes, and during which time twelve heats were made in one furnace and 11 heats were made in the other, with a total tonnage for the two furnaces of 1,615 tons for the week.

The conditions under which the data for this test was obtained, were strictly operating conditions, and the heat balance is figured from the average of all collected data.

The log of the test follows:

FURNACE CHARGES

Furnaces	No. 89 Pounds	No. 90 Pounds	Total Pounds
Charges:			
Hot metal	938,800	871,200	1,810,000
Cold pig	117,000	38,600	155,600
Blooms	474,500	479,200	953,700
Plate	.....	31,500	31,500
Ingot butts	55,000	40,600	95,600
Rail scrap	25,500	38,000	63,500
Tin scrap	43,000	.....	43,000
Turnings	5,500	.....	5,500
Steel scrap	86,900	81,000	167,900
Stirring rods	2,400	2,200	4,600
Additions:			
Hot metal	122,300	143,900	266,200
Ferro manganese	8,600	5,300	13,900
Ferro silicon	650	400	1,050
Roll sulphur	245	120	365
Ore charged	185,500	139,100	324,600
Ore added	61,000	39,000	100,000
Cinder charged	5,000	4,100	9,100
Scale charged	8,900	.....	8,900
Limestone	126,100	122,400	248,500
Fluor spar	2,250	2,200	4,450
Dolomite	57,000	53,000	110,000
Coal (anthracite slack)	1,300	2,150	3,450
Pounds product	1,905,450	1,713,040	3,618,490
Ingots made	309	271	580
Heats made	12	11	23
Average analysis:			
C	0.320	0.462	0.388
Mn	0.506	0.484	0.495
S	0.0426	0.0430	0.0428
P	0.0174	0.0308	0.0238
Si	.....	0.041	0.0200

The starting point in figuring the distribution of heat in an open-hearth furnace is naturally the fuel, hence the following log was necessary in order to get the time of operation of the Gas Producers. The logs of the operation of furnaces 89 and 90, as here given, put the gas producer operation on a test basis, and which summated gives the steam blown in the operation. Summating the data in relation to the amount of gas used during the operation of the furnaces, and taking into consideration the time the producers were off for cleaning, we get the table showing the steam blown.

OPEN-HEARTH FURNACE, NO. 89

Date	Hr. Min.	Remarks
Saturday		
20th	11:25 A.M.	Commenced charging for a soaker.
20th	7:26 P.M.	Gas producers started. Steam on.
20th	8:00 P.M.	Ended charging scrap for soaker. Gas on.
21st	6:45 A.M.	Hot metal added. Gas lowered.
21st	2:30 P.M.	Commenced tapping heat, No. 89,476. No. 1.
21st	2:43 P.M.	Finished tapping heat, No. 89,476. Start patch.
21st	2:50 P.M.	Finished patching bottom. Gas off.
21st	3:02 P.M.	Gas on again.
21st	3:15 P.M.	Commenced charging for heat, No. 89,477.
21st	6:15 P.M.	Finished charging.
22d	4:57 A.M.	Commenced tapping. Heat No. 89,477. No. 2.
22d	5:10 A.M.	Finished tapping.
22d	5:25 A.M.	Gas taken off furnace.
22d	5:51 A.M.	Gas put on again.
22d	8:07 A.M.	Commenced charging for heat No. 89,478. No. 3.
22d	8:40 A.M.	Finished first charge.
22d	11:25 A.M.	Hot metal to furnace. Gas lowered.
22d	11:40 A.M.	Gas increased.
22d	5:50 P.M.	Tapping commenced on heat No. 89,478. No. 3.
22d	5:55 P.M.	Tapping finished, making bottom. Gas off.
22d	7:05 P.M.	Bottom made. Gas put on again.
22d	7:30 P.M.	Commenced charging for heat No. 89,479. No. 4.
22d	8:30 P.M.	Finished charging.
23d	4:45 A.M.	Hot metal added. Gas lowered.
23d	4:49 A.M.	Commenced tapping heat No. 89,479. No. 4.
23d	5:15 A.M.	Gas taken off, tapping finished. Make bottom.
23d	6:00 A.M.	Gas put on again. Bottom made.
23d	6:25 A.M.	Commenced charging for heat No. 89,480. No. 5.
23d	9:17 A.M.	Hot metal added. Gas lowered.
23d	9:30 A.M.	Finished charging. Gas increased.
23d	4:22 P.M.	Commenced tapping heat No. 89,480. No. 5.
23d	4:32 P.M.	Finished tapping. Gas taken off.
23d	5:20 P.M.	Gas put on again. Bottom made.
23d	5:40 P.M.	Commenced charging for heat No. 89,481. No. 6.
23d	8:10 P.M.	Finished charging.
24th	3:32 A.M.	Commenced tapping heat No. 89,481. No. 6.
24th	3:40 A.M.	Tapping finished. Bottom making. Gas off.
24th	4:06 A.M.	Gas on again. Bottom made.
24th	4:15 A.M.	Commenced charging for heat No. 89,482. No. 7.
24th	6:10 A.M.	Hot metal added. Gas lowered.
24th	6:15 A.M.	Charging completed. Gas increased.
24th	4:15 P.M.	Commenced tapping heat No. 89,482. No. 7.
24th	4:25 P.M.	Tapping complete, Bottom making. Gas off.
24th	5:10 P.M.	Bottom made. Gas put on again.
24th	7:07 P.M.	Commenced charging for heat No. 89,483. No. 8.
24th	9:05 P.M.	Charging finished.
25th	4:40 A.M.	Commenced tapping heat No. 89,483. No. 8.
25th	4:50 A.M.	Tapping complete. Making bottom. Gas off.
25th	5:30 A.M.	Bottom made. Gas put on again.
25th	8:35 A.M.	Commenced charging for heat No. 89,484. No. 9.
25th	11:00 A.M.	Hot metal added. Gas lowered.
25th	11:15 A.M.	Gas increased. Hot mteal in. Charge all in.
25th	4:55 P.M.	Tapping commenced on heat No. 89,484. No. 9.
25th	5:10 P.M.	Tapping completed. Gas off. Bottom.
25th	5:45 P.M.	Bottom made. Gas put on again.
25th	6:55 P.M.	Commenced charging for heat No. 89,485. No. 10.
25th	8:30 P.M.	Finished charging.
26th	3:41 A.M.	Commenced tapping heat No. 89,485. No. 10.
26th	3:53 A.M.	Tapping finished. Gas off. Bottom making.
26th	4:40 A.M.	Gas on again. Bottom made.
26th	6:15 A.M.	Commenced charging for heat No. 89,486. No. 11.
26th	8:38 A.M.	Hot metal added. Gas lowered.
26th	8:50 A.M.	Charging completed. Gas increased.
26th	4:40 P.M.	Commenced tapping heat No. 89,486. No. 11.
26th	4:50 P.M.	Tapping completed. Gas off. Making bottom.
26th	5:43 P.M.	Bottom made. Gas on again.
26th	6:07 P.M.	Commenced charging for heat No. 89,487. No. 12.
26th	8:20 P.M.	Charging completed.
27th	3:35 A.M.	Hot metal added. Gas lowered and increased.
27th	4:45 A.M.	Commenced tapping heat No. 89,487. No. 12.
27th	4:58 A.M.	Tapping completed. Gas off. Making bottom.
27th	6:00 A.M.	Bottom made. Gas on again low. Increased.
27th	9:35 A.M.	Cold pig charged for soaker.
27th	12:58 P.M.	End of test. Steam off producers.

OPEN-HEARTH FURNACE. NO. 90

Date	Hr.	Min.	Remarks
Saturday			
20th	7	26 P.M.	Gas producers started. Steam on.
20th	8	30 P.M.	Furnace still empty. Gas on, low.
21st	1	45 A.M.	More gas used.
21st	3	15 A.M.	Gas again increased.
21st	8	15 A.M.	Gas taken off.
21st	8	55 A.M.	Gas put on again.
21st	10	45 A.M.	Charging cold pig. Charging commenced.
21st	11	25 A.M.	Gas put on full.
21st	3	25 P.M.	Charging complete for heat No. 90,500. No. 1.
21st	10	25 P.M.	Commenced tapping heat No. 90,500. No. 1.
21st	10	40 P.M.	Finished tapping. Making bottom. Gas off.
21st	11	20 P.M.	Bottom made. Gas on again.
22d	12	32 A.M.	Commenced charging for heat No. 90,501. No. 2.
22d	12	57 A.M.	Gas increased.
22d	3	15 A.M.	Hot metal added. Gas lowered.
22d	3	30 A.M.	Charging completed. Gas increased.
22d	9	30 A.M.	Hot metal added. Gas lowered.
22d	9	41 A.M.	Commenced tapping heat No. 90,501. No. 2.
22d	9	54 A.M.	Tapping complete. Making bottom. Gas off.
22d	10	12 A.M.	Making bottom complete. Gas on again.
22d	11	25 A.M.	Commenced charging for heat No. 90,502. No. 3.
22d	2	24 P.M.	Gas increased more.
22d	3	45 P.M.	Hot metal added. Gas lowered.
22d	4	00 P.M.	Gas increased.
22d	10	36 P.M.	Commenced tapping. Heat No. 90,502. No. 3.
22d	10	50 P.M.	Tapping complete. Gas still on.
22d	12	15 A.M.	Gas off. Making bottom.
22d	12	57 A.M.	Bottom made. Gas on again.
23d	1	00 A.M.	Commenced charging. Heat No. 90,503. No. 4.
23d	2	30 A.M.	Gas increased.
23d	2	40 A.M.	Scrap charged.
23d	4	55 A.M.	Hot metal added. Gas lowered.
23d	5	00 A.M.	Charge completed.
23d	2	30 P.M.	Hot metal added.
23d	2	34 P.M.	Tapping commenced. Gas still high.
23d	2	50 P.M.	Tapping complete. Gas off. Bottom make.
23d	3	13 P.M.	Bottom made. Gas put on.
23d	3	38 P.M.	Commenced charging for heat No. 90,504. No. 5.
23d	5	55 P.M.	Charging complete.
24th	4	35 A.M.	Commenced tapping heat No. 90,504. No. 5.
24th	4	50 A.M.	Tapping complete. Gas still on.
24th	6	00 A.M.	Making bottom. Gas taken off.
24th	7	05 A.M.	Gas on again. Bottom made.
24th	9	30 A.M.	Gas increased.
24th	10	52 A.M.	Gas taken off again.
24th	10	56 A.M.	Gas put on again.
24th	11	00 A.M.	Charging commenced for heat No. 90,505. No. 6.
24th	1	37 P.M.	Hot metal added. Gas lowered.
24th	1	45 P.M.	Charging complete. Gas increased.
24th	10	20 P.M.	Tapping commenced.
24th	10	30 P.M.	Tapping complete. Gas off. Bottom.
24th	10	38 P.M.	Bottom made. Gas on again.
24th	11	45 P.M.	Commenced charging for heat No. 90,506. No. 7.
25th	2	10 A.M.	Charging complete.
25th	2	45 A.M.	Gas increased.
25th	4	15 A.M.	Gas increased.
25th	4	55 A.M.	Gas decreased.
25th	9	27 A.M.	Commenced tapping heat No. 90,506. No. 7.
25th	9	35 A.M.	Tapping finished. Gas off. Making bottom.
25th	9	50 A.M.	Tapping and bottom made. Gas on again.
25th	10	05 A.M.	Charging commenced for heat No. 90,507. No. 8.
25th	11	20 A.M.	First charge complete.
25th	1	15 P.M.	Charging complete.
25th	11	40 P.M.	Hot metal added. Gas still high.
25th	11	52 P.M.	Finished tapping. Gas still on. No. 8.
25th	12	00 A.M.	Making bottom. Gas off.
25th	12	55 A.M.	Bottom made. Gas put on again.
26th	2	23 A.M.	Commenced charging heat No. 90,508. No. 9.
26th	4	20 A.M.	Charging completed. Gas increased.
26th	1	54 P.M.	Commenced tapping heat No. 90,508. No. 9.
26th	2	05 P.M.	Tapping finished. Gas off. Bottom.
26th	2	36 P.M.	Bottom made. Gas put on again.
26th	2	39 P.M.	Commenced charging heat No. 90,509. No. 10.
26th	4	15 P.M.	Finished charging.
27th	1	07 A.M.	Commenced tapping No. 90,509. No. 10.
27th	1	15 A.M.	Finished tapping.
27th	1	32 A.M.	Making bottom. Gas off.
27th	2	02 A.M.	Bottom made. Gas on again.
27th	2	09 A.M.	Commenced charging No. 90,510. No. 11.
27th	4	48 A.M.	Hot metal added. Gas lowered.
27th	4	52 A.M.	Gas increased.
27th	12	35 P.M.	Commenced tapping No. 90,510. No. 11.
27th	12	52 P.M.	Tapping, etc., completed.
27th	12	58 P.M.	Producers down end of test.

TIME PRODUCERS WERE OFF FOR CLEANING

DATE	PRODUCERS	No. 1	2	3	4
21	Minutes	25	30	40 70	25
22		30	35	..	35
23		30	30	50	30
24		55	35	..	30
25		55	35	55	30
26		60	40	55	..
27		..	..	..	30

It will be noted in the table for steam consumed that a total figure for steam blown is obtained, this being the figure that is often used in gas producer testing. When the producers are operating at one-half, one-fourth, or no load, less steam is used and in direct proportion to the gas used, hence the above deductions, giving the net quantity of steam consumed.

STEAM BLOWN

TEST OF 161 HOURS, 32 MINUTES, ON FOUR PRODUCERS AND TWO FURNACES

Furnace	Charging	Working	Hot Metal	Tapping	Patching	Waiting on Material
Gas On	On ¾	On Full	On ½	On Full	Off	On ½
No. 89	23° 33′	106° 26′	1° 19′	2° 33′	8° 29′	19° 12′
No. 90	33° 22′	92° 12′	1° 17′	2° 15′	6° 11′	26° 36′

Steam nozzle, 0.4375″ diameter, hence 0.1500 square inches area.
Average gauge pressure when working = 45 pounds..... = 60 lbs. absolute
Steam per minute per producer = (Area × Abs. P. ÷ 70) × 60 = 7.713 pounds
Total steam, no delays, for four producers for 161° 32′.... = 299,018 pounds
Delays:

Equivalent of one producer off cleaning for.....	15°	10′ =	7,019 pounds
Equivalent of two producers off when patching..	14°	40′ =	13,575 pounds
Gas on ¾ when adding hot metal.....	2°	36′ =	1,805 pounds
Two producers at ½ steam for.....	45°	48′ =	21,195 pounds
Two producers at ¾ steam for.....	56°	55′ =	13,168 pounds
Total.....			56,762 pounds

Steam used during the test = 299,018 − 56,762..... = 242,256 pounds

TABLE NO. 1

	Per Cent.	Weight, Pounds	Per Ton of Coal	Per Ton of Ingots, 1615
Coal charged.....	100.00	856,000	2,240	530 pounds
Ash loss.....	5.65	48,360	126.5	29.93
Coal lost in ash.....	1.07	9,160	23.9	5.65
Coal lost in soot.....	.69	5,906	15.4	3.64
Total coal loss.....	1.76	15,066	39.3	9.29
Moisture.....	1.11	9,500	24.8	5.86
Coal gasified.....	98.24	840,934	2,200.7	520.0
Carbon in coal gasified.....	76.23	641,000	1,677.5	396.0
Carbon in gas, per cubic foot.....	.....	(0.010802)	.....	.....
Cubic feet gas made..	.....	59,340,000	155,290	36,740. cu. ft.
Pounds of gas made..	.....	4,219,000	11,040	2,612. pounds
Steam blown.....	.....	242,256	634	150.3
Air blown.....	.....	3,353,180	8,775	2,075

Table I shows the distribution of carbon from the coal in the gas made and in the refuse produced in the process. This table shows the distribution per ton of coal and also per ton of ingots made, and is taken from test conditions, and during this test some 530 lb. of coal were required per ton of ingots made. Since it is desired to have this heat balance figured for strictly operating conditions, the figure 510 lb. of coal per ton of ingots is taken, this being an average figure, and the results figured accordingly. Table II is thus figured from the results obtained from a test, and from which the conversion of one ton of coal into gas is obtained, and these figures used in obtaining the distribution for the rate of 510 lb. of coal per ton of ingots made.

TABLE NO. 2

	Per Cent.	Weight	Per Ton Coal	Per Ton Ingots
Coal charged.....	100.00	829,000	2,240.	510.lbs
Ash loss.....	5.65	46,840	126.5	28.8
Coal loss in ash.....	1.07	8,870	23.9	5.5
Coal loss in soot.....	.69	5,720	15.4	3.5
Total coal loss.....	1.76	14,590	39.3	9.0
Moisture.....	1.11	9,200	24.8	5.66
Coal gasified.....	(98.24)	814,410	2,200.7	501.
Carbon in coal gasified...	(76.23)	620,800	1,682.4	381.7
Carbon per cubic foot of gas.....	0.010802	.....	.....	.....
Cubic feet of gas made...	.....	57,400,000	155,100.	35,310.
Pounds of gas made (dry).....	.....	4,081,000	11,028.	2,510.
Steam blown.....	.....	242,256	654.	149.
Air blown.....	.....	3,289,000	8,888.	2,024.

Following is the analysis of the coal. The proximate analysis is representative of that obtained in the laboratory. Some difficulty was experienced in obtaining the ultimate analysis, especially considering the hydrogen, but the ultimate analysis here given is obtained by Dr. Parr's method and the hydrogen content assured of.

ANALYSIS OF COAL

Proximate Analysis		Ultimate Analysis of Volatile	
Volatile matter	34.94%	Carbon	17.93%
Fixed carbon	58.30%	Hydrogen	5.02%
Total carbon	76.23%	Nitrogen	1.38%
Ash	5.65%	Oxygen	10.61%
Moisture	1.11%	Total	34.94%
Combustible in ash	15.13%		
Pure coal	93.24%		
Coal lost in the ash. Pure coal = 93.24%		True ash	= 84.87%
Ash	= 5.65%	Combustible	= 15.13%
Moisture	= 1.11%		
Refuse produced in burning coal. = 5.65 + 84.87%			= 6.65%
6.65 − 5.65 = 1.00 = Combustible. 1.00 + 93.24%			= 1.07%
1.07% = Coal lost in ash.			

Approximate Analysis of Ash, Estimated

Fe	= .85
S	= 1.16
SiO_2	= 1.54
CaO	= .80
Al_2O_3	= 1.30
Total	= 5.65%

In the following analysis of gases, the gases are figured at 32 deg. F., and not at 62 deg. F., as is common practice, for the reason that figuring at 32 deg. F simplifies the transfer to the metric system. It would be a great advantage in many calculations if 32 deg. F., and not 62 deg. F., were taken as the standard temperature for measuring gases.

GAS PRODUCERS. ANALYSIS OF GAS
VOLUME CONVERTED INTO WEIGHT. GAS AT 32°F.

	Volume	Standard Weights	By Weight at 32°F.	Standard BTU, 32°	BTU, in Gas 32°	BTU, in Gas 62°	Weight of Gas at 32°
CO_2	5.905	0.12424	10.317	.....	.....	.....	0.007336
CO	22.625	0.07858	25.003	3.49	78.961	.....	0.017778
O_2	0.020	0.08953	0.025	.....	.....	.....	0.000018
CH_4	2.700	0.04505	1.711	10.65	28.755	.....	0.001216
C_2H_4	0.415	0.07870	0.458	16.73	6.943	.....	0.000326
H_2	12.580	0.00565	1.000	3.48	43.778	.....	0.000711
N_2	55.755	0.07859	61.486	.....	.....	.....	0.043718
Total	100.000	.....	100.000	.....	158.437	149.740	0.071103

ANALYSIS OF GAS

Ultimate Analysis		Carbon in the Gas. 32°F.	Analysis of Air	
			Wet	Dry
C	15.206	Carbon in CO_2 = 0.002001	N_2 ..... 75.92	77.00
O_2	21.815	CO = 0.007610	O_2 ..... 22.67	23.00
N_2	61.486	CH_4 = 0.000912	H_2O 1.41	
H_2	1.493	C_2H_4 = 0.000279		
Total	100.000	Total ..... = 0.010802		

Weight of gas per cubic foot, at 32°F ..... = 0.071103 pound
Weight of carbon per cubic foot at 32°F ..... = 0.010802 pound

ELEMENTARY DISTRIBUTION AT RATES OF 510 AND 530 POUNDS COAL-TON INGOTS

510. Rate	Weight	%	C.	%	O_2	%	N_2	%	H_2
Coal	814,410	76.23	620,800	10.61	86,410	1.38	11,240	5.02	40,890
Air	3,289,000	...	.....	22.67	745,800	75.92	2,497,260	...	.....
Steam	180,360	...	.....	88.88	160,320	...	.....	11.1	20,040
Gas	4,081,000	15.21	620,800	21.81	890,200	61.48	2,508,500	1.49	60,930

530. Rate	Weight	C	O_2	N_2	H_2
Coal	840,934	641,000	89,220	11,600	42,210
Air	3,353,180	.....	770,780	2,582,400	.....
Steam	187,090	.....	166,300	.....	20,790
Gas	4,219,000	641,000	920,400	2,594,000	63,000

MOISTURE DISTRIBUTION

Rate		530 lb.	510 lb.
Moisture in coal	1.11	9,500	9,200
Moisture in air	1.41	47,280	46,380
Moisture in steam	100.	242,256	242,256
Total moisture		299,036	297,836
Decomposed		187,090	180,360
Free moisture		111,946	117,476
Grains		783,600,000	822,200,000
Cubic feet gas made		59,340,000	57,400,000
Grains moisture per cubic foot		13.2	14.3
B.T.U., per cubic foot, gas 32° F.		158.4	158.4
B.T.U. in gas (latent heat)		9,399,456,000	9,070,000,000

Having figured the fuel down to a working basis, thereby obtaining the heat available and the losses in the making of the gas, it is necessary to similarly treat the materials charged into the furnace. On account of the numerous chemical reactions involved it is necessary to compile an "elementary distribution," with the object in view of their final distribution and arrangement in the metal, slags, and gases. Accordingly, the following analyses have been taken as representative of the materials charged, and for the object in view have been adjusted to read 100 per cent, since no laboratory analyses are reported as totaling 100 per cent.

OPEN HEARTH. ANALYSIS OF RAW MATERIALS
ANALYSES ADJUSTED TO READ 100%

Element	Cold Pig Hot Metal	Ferro Manganese	Ferro Silicon	Blooms	Plate	Ingot Butts	Rail Scrap
Si	1.05	0.70	52.00	.025	.....	.025	.15
Mn	1.48	78.00	.....	.500	.350	.500	.90
C	3.41	5.75	0.20	.400	.250	.400	.47
P	.294	.325	.050	.030	.040	.030	.080
S	.038	.155	.005	.040	.040	.040	.050
Fe	93.728	15.070	47.745	96.005	95.820	95.005	97.350
Fe_3O_4	.....	.....	.....	3.000	3.500	4.000	1.000

Element	Tin Scrap	Turnings	Steel Scrap	Stirring Rods	Anthracite Slack
Si	.....	.020	.020	.020	VM. = 7.45
Mn	.40	.480	.500	.450	FC. = 75.56
C	.18	.180	.250	.150	Ash. = 16.99
P	.075	.045	.035	.035	TC. = 75.56 + 3.72
S	.040	.045	.035	.035	O. = 2.38
Fe	97.305	99.210	96.160	96.310	N. = .40
Fe_3O_4	2.000	.000	3.000	3.000	H. = 1.05

Element	Roasted Magnesite	Calcined Dolomite		Standard Limestone	Fluor Spar	Soft Ore Charged	Lump Ore	Roll Scale
SiO_2	2.66	.22	.403	0.49	0.50	6.290	3.71	1.44
Fe_2O_3	7.43	1.43	2.637	1.43	0.30	70.148	93.456	.....
Al_2O_3	1.46	.26	.476	0.26	0.40	2.350	1.57	.....
MnO	.40	.....	.....	.....	0.00	0.600	.117	0.40
CaO	4.18	31.34	57.400	55.15	....	0.200	.490	.....
P_2O_5	.066	.004	.007	.005	....	.152	.037	.158
$CaFl_2$	.....	.....	.....	.....	95.00	.....	.....	.....
MgO	83.680	21.30	38.897	.33	....	.....	.200	.....
Moisture	0.124	.10	.180	.30	3.80	20.260	.420	3.100
Fe_3O_4	.....	.....	.....	.....	....	....	....	94.902
CO_2	.....	45.346	.....	42.035	....	....	....	.....

The following temperatures are the results of averaging many observations, and can be taken as representative of the temperatures met with in the parts of the furnace quoted. This table also shows the efficiency of the producers and the regenerators, which figures are obtained in the heat balance which follows.

SUMMARY
MATERIALS USED PER TON OF INGOTS PRODUCED:

Producer Gas:		
Coal	510.00	
Steam	149.00	
Air	1,995.50	H_2O in gas (downtake) = 72.00
H_2O (air)	28.50	H_2O in gas (valve) = 96.80
H_2O (coal)	5.60	
Gas made (dry)	2,510.00	
H_2O (gas)	72.00	
Hot gas efficiency of the producers		94.99%
Efficiency at delivery to the regenerator entrance		91.77%
Temperature of gas regenerator		2,120°F.
Temperature of gas coming from regenerators		1,940°F.
Temperature of gas going to regenerators		1,100°F.
Pounds of air required for combustion of producer gas		7,378.00
Moisture in this air		106.00
Temperature of air regenerator		2,260°F.
Temperature of air going to regenerator		90° F.
Temperature of air coming from regenerator		2,012°F.
Efficiency of the regenerators		79.22%
Reversals on No. 90 Furnace, during the week's test:		
Total number during week	= 509.	
Average time	= 19.0 minutes	
Longest time	= 70.0 minutes	
Shortest time	= 2.0 minutes	
Temperature of steel running from furnace	= 3,012°F.	temperature of stee
Temperature of burned gases at end of furnace	= 2,832°F.	
Temperature of gases just outside regenerators	= 1,252°F.	
Temperature of stack	= 1,200°F.	

TABLE 3-A

TABLE OF DISTRIBUTION OF THE VARIOUS CONSTITUENTS OF THE CHARGE AND THEIR SUBSEQUENT CHEMICAL ARRANGEMENT IN THE FURNACE—NAMELY METAL, SLAG AND GASES

Material	Pounds	Si		SiO_2		Mn		MnO		P		As. P_2O_5		Fe		Fe_2O_3	
		%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal	1,810,000	1.050	19,000	....	40,720	1.480	26,790	....	....	.294	5,322.	....	....	93.728	1,696,550	....	.....
Cold pig	155,600	1.050	1,634	....	3,502	1.480	2,303	....	....	.294	457.	....	....	93.728	145,800	....	.....
Bloom	953,700	.025	238	....	510	.500	4,769	....	....	.030	286.	....	....	96.005	915,650	....	.....
Plate	31,500	....	....	....	....	.350	110	....	....	.040	13.	....	....	95.820	30,185	....	.....
Ingot butts	95,600	.025	24	....	51	.500	478	....	....	.030	28.	....	....	95.005	90,820	....	.....
Rail scrap	63,500	.150	95	....	204	.900	571	....	....	.080	51.	....	....	97.350	61,810	....	.....
Tin plate	43,000	....	....	....	....	.400	172	....	....	.075	32.	....	....	97.305	41,845	....	.....
Turnings	5,500	.020	1	....	2	.480	26	....	....	.045	3.	....	....	99.210	5,457	....	.....
Steel scrap	167,900	.020	34	....	72	.500	839	....	....	.035	58.	....	....	96.160	161,450	....	.....
Stirring rods	4,600	.020	1	....	2	.450	21	....	....	.035	2.	....	....	96.310	4,430	....	.....
Total	3,330,900	.631	21,027	....	45,063	1.084	36,079	....	46,576	.187	6,251.	....	14,313	94.695	3,153,997	....	.....
Fine ore	324,600	....	....	6.290	20,420	....	....	.600	1,947	...	.....	.152	493	....	.....	70.148	227,700
Cinder	9,100	....	....	5.100	464	....	....	.400	37	...	.....	.160	15	....	.....	....	.....
Scale	8,900	....	....	1.440	128	....	....	.400	35	...	.....	.158	14	....	.....	....	.....
Dolomite	110,000	....	....	.403	443	....	....	....	....	...	.....	.007	8	....	.....	2.637	2,900
Limestone	248,500	....	....	.490	1,217	....	....	....	....	...	.....	.005	12	....	.....	1.430	3,554
Lump ore	100,000	....	....	3.710	3,710	....	....	.117	117	...	.....	.037	37	....	.....	93.456	93,456
Fluor spar	4,450	....	....	.300	22	....	....	....	....	...	.....	....	....	....	.....	.300	14
Total, added	805,550	....	....	3.277	26,404	....	....	.265	2,136	...	.....	.0717	578	....	.....	40.670	327,623
Run off slag	109,084	....	....	10.020	10,930	....	....	9.230	10,067	...	.....	1.998	2,180	27.825	30,353	....	.....
Normal slag run	260,379	....	....	15.500	40,358	....	....	7.915	20,600	...	.....	2.845	7,408	14.980	39,000	....	.....
Hot metal	266,200	1.050	2,795	....	5,990	1.480	3,940	....	....	.294	782.5	....	....	93.728	249,500	....	.....
Fe manganese	13,900	.700	97	....	208	78.000	10,840	....	....	.325	45.	....	....	15.070	2,095	....	.....
Fe silicon	1,050	52.000	546	....	470	....	....	....	....	.050	.5	....	....	47.745	501	....	.....
Sulphur	365	....	....	....	....	....	....	....	....	....	....	....	....	....	.....	....	.....
Coal	3,450	....	....	7.000	241	....	....	....	....	....	.....	....	....	....	.....	....	.....
Aluminum	411	....	....	....	....	....	....	....	....	....	.....	....	....	2.000	69	....	.....
Total, No. 3	285,376	1.205	3,438	2.826	7,609	5.180	14,780	....	....	.290	828.	....	....	88.340	252,165	....	.....
Furnace steel	3,345,087	....	....	....	....	.179	6,000	....	7,745	.0209	700.	....	1,600	99.640	3,334,097	....	.....
Product steel	3,618,490	.020	724	....	1,551	.495	17,910	....	....	.0238	861.	....	....	....	3,586,000	....	.....
Difference	276,823	....	....	....	1,551	....	11,910	....	....	....	161.	....	....	....	.....	....	.....
In recarburizer	285,376	....	....	2.826	7,609	5.180	14,780	....	....	.290	828.	....	....	....	.....	....	.....
To ladle slag	12,705	....	....	47.680	6,058	22.590	2,870	29.060	3,692	5.250	667.	11.150	1,468	....	.....	....	.....
One-third normal slag	130,189	....	....	15.500	20,179	....	....	7.915	10,300	...	.....	....	3,703	....	.....	....	.....
Ladle slag	142,894	....	....	18.360	26,237	....	....	8.792	13,992	...	.....	3.620	5,171	....	.....	....	.....
Ladle gases	.....	....	....	....	....	....	....	....	....	...	.....	....	....	....	.....	....	.....
Producer gas	4,198,476	....	....	....	....	....	....	....	....	...	.....	....	....	....	.....	....	.....
Air used	12,090,000	....	....	....	....	....	....	....	....	...	.....	....	....	....	.....	....	.....
Flue gases	17,104,426	....	....	....	....	....	....	....	....	...	.....	....	....	....	.....	....	.....

TABLE 3-B

Material	Fe_3O_4		As. FeO		Fe From Oxides	Oxygen		Carbon		CO_2		Sulphur		Aluminum	
	%	Lbs.	%	Lbs.		Absorbed	Freed	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal	.....	....	....	....	.....	.....	.....	3.410	61,720	....	.....	.038	687	...	...
Cold pig	.....	....	....	....	.....	.....	.....	3.410	5,306	....	.....	.038	59	...	...
Bloom	3.000	28,600	....	....	.....	.....	.....	.400	3,815	....	.....	.040	381	...	...
Plate	3.500	1,102	....	....	.....	.....	.....	.250	79	....	.....	.040	13	...	...
Ingot butts	4.000	3,824	....	....	.....	.....	.....	.400	382	....	.....	.040	38	...	...
Rail scrap	1.000	635	....	....	.....	.....	.....	.470	298	....	.....	.050	32	...	...
Tin plate	2.000	860	....	....	.....	.....	.....	.180	78	....	.....	.040	17	...	...
Turnings	.....	....	....	....	.....	.....	.....	.180	10	....	.....	.045	3	...	...
Steel scrap	3.000	5,036	....	....	.....	.....	.....	.250	419	....	.....	.035	59	...	...
Stirring rods	3.000	138	....	....	.....	.....	.....	.150	69	....	.....	.035	2	...	...
Total	1.207	40,195	....	....	29,106 (Fe_3O_4)	42,595	11,089	3.164	72,076	....	.....	.0387	1,290	...	...
Fine ore	.....	....	....	....	.....	.....	}	.....	.....	....	.....	...	...	...	...
Cinder	94.340	8,584	....	....	.....	.....	} 104,808	.....	.....	....	.....	...	...	...	...
Scale	94.902	8,448	....	....	.....	.....	}	.....	.....	....	.....	...	...	...	...
Dolomite	....	....	....	....	.....	.....	.....	.....	.....	....	.....	...	...	...	...
Limestone	.....	....	....	....	.....	.....	75,965	.....	28,485	42.035	104,450	...	...	...	...
Lump ore	.....	....	....	....	.....	.....	.....	.....	.....	....	.....	...	...	...	...
Fluor spar	.....	....	....	....	229,331 (Fe_2O_3)	.....	.....	.....	.....	....	.....	...	...	...	...
Total, added	2.110	17,032	....	....	10,516 (Fe_3O_4)	.....	.....	.....	.....	12.965	104,450	...	...	...	...
Run off slag	.....	....	35.750	39,000	.....	8,647	.....	.....	.....	....	.....	...	...	...	...
Normal slag run	.....	....	19.250	50,117	.....	16,675	.....	.....	.....	....	.....	...	...	...	...
Hot metal	.....	....	....	....	.....	.....	.....	3.410	9,077	....	.....	.038	101	...	...
Fe manganese	.....	....	....	....	.....	.....	.....	5.750	799	....	.....	.155	21	...	...
Fe silicon	.....	....	....	....	.....	.....	.....	.200	2	....	.....	.005	...	...	...
Sulphur	.....	....	....	....	.....	.....	.....	.....	.....	....	.....	100.0	365	...	...
Coal	.....	....	....	....	.....	.....	59	79.280	2,735	....	.....	...	...	...	...
Aluminum	.....	....	....	....	.....	.....	.....	.....	.....	....	.....	...	...	100	411
Total, No. 3	.....	....	....	....	.....	.....	.....	4.420	12,613	....	.....	...	487	...	...
Furnace steel	.....	....	....	....	3,334,097	2,669	.....	.0896	3,000	....	.....	.0385	1,290	...	...
Product steel	.....	....	....	....	3,586,000	.....	.....	.3880	14,040	....	.....	.0428	1,548	...	...
Difference	.....	....	....	....	251,903	.....	.....	.....	11,040	....	.....	...	258	...	...
In recarburizer	.....	....	....	....	252,165	4,396	.....	.....	12,613	....	.....	...	487	...	...
To ladle slag	.....	....	2.630	335	262	.....	1,695	.....	.....	....	.....	...	...	...	...
One-third normal slag	.....	....	....	25,058	.....	.....	.....	.....	.....	....	.....	...	...	...	...
Ladle slag	.....	....	17.770	25,393	.....	.....	.....	.....	.....	....	.....	...	...	...	...
Ladle gases	.....	....	....	....	.....	.....	59	.....	1,573	....	.....	...	229	...	...
Producer gas	.....	....	....	....	.....	21.200%	890,200	14.780	620,800	....	.....	...	...	...	...
Air used	.....	....	....	....	.....	22.670%	2,741,000	.....	.....	....	.....	...	...	...	...
Flue gases	.....	....	....	....	.....	21.967%	3,757,790	4.200	718,361	....	.....	...	...	...	...

TABLE 3-C

Material	Al_2O_3		CaO		$CaFl_2$		MgO		H_2		N_2		Moisture	
	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal														
Cold pig														
Bloom														
Plate														
Ingot butts														
Rail scrap														
Tin plate														
Turnings														
Steel scrap														
Stirring rods														
Total														
Fine ore	2.350	7,628	.200	649									20.260	65,760
Cinder														
Scale													3.100	276
Dolomite	.476	524	57.400	63,140			38.897	42,790					.180	198
Limestone	.260	646	55.150	137,050			.330	820					.300	745
Lump ore	1.570	1,570	.490	490			.200	200					.420	420
Fluor spar	.400	18			95.000	4,227							3.800	169
Total, added	1.289	10,386	24.990	201,329	.524	4,227	5.438	43,810					8.386	67,568
Run off slag	3.006	3,279	33.985	37,070			6.012	6,558						
Normal slag run	1.819	1,738	42.050	109,506	1.083	2,818	9.540	24,935						
Hot metal														
Fe manganese														
Fe silicon														
Sulphur														
Coal	6.000	207	1.990	68					1.00	34	1.00	34		
Aluminum		877												
Total, No. 3		1,084	.535	68						34		34		
Furnace steel														
Product steel														
Difference														
In recarburizer														
To ladle slag	8.530	1,084	.535	68										
One-third normal slag		2,369		54,753		1,409		12,417						
Ladle slag	2.417	3,453	38.370	54,821	.986	1,409	8.690	12,417						
Ladle gases														
Producer gas									1.451	60,930	59.75	2,508,500	2.798	117,476
Air used											75.92	9,176,861	1.410	170,500
Flue gases											68.33	11,685,361	5.514	942,914

Taking each of the materials charged, their weight and analysis, the tables 3-A, 3-B and 3-C are obtained, which materials split up into the various oxides formed, and also those reduced, together with other reactions, give the elementary ically, from the elementary distribution in tables 3-A, -B, -C, and also gives a check analysis, obtained from actual analysis in practice. This table also gives the analysis of the flue gases, both wet and dry, by volume and by weight.

TABLE IV

PRODUCER GAS DISTRIBUTION. (WET ANALYSIS BY WEIGHT)

Gas	Pounds	Oxygen		Carbon		Hydrogen		Nitrogen		Moisture	
		%	Weight	%	Weight	%	Weight	%	Weight	%	Weight
Producer gas, downtake	4,198,476	21.20	890,200	14.78	620,800	1.451	60,930	59.750	2,508,500	2.791	117,476
At reversing valve	4,237,476	21.01	890,200	14.65	620,800	1.438	60,930	59.200	2,508,500	3.693	156,476
From furnace gases add			176,590		97,561						67,568
Combustion of H_2 add											548,370
Air used	12,090,000	22.67	2,741,000					75.920	9,176,861	1.410	170,500
Flue gases made	17,104,426	21.967	3,757,790	4.200	718,361			68.330	11,685,361	5.514	942,914

final arrangement in the metal, slags, and gases. It will be noted that a run-off slag is first produced, then a normal slag, a portion of which normal slag goes into the ladle to help make up the ladle slag.

The above Table IV on the producer gas is here given to show the distribution of moisture. It will be noted that there is an increase of 1 per cent in the amount of moisture carried in the producer gas after it has passed the reversing valve. Reversing valves, in nearly all cases are water cooled, and the hot producer gas coming in contact with a water seal naturally picks up a quantity of moisture, which throws a large portion of sensible heat that was of service into latent heat that cannot be used. Does not this increase in moisture, and the accompanying decrease in the effective heat delivered to the furnace, permit of the suggestion that reversing valve design might be improved, with the use of other material, such as tar, as a sealing fluid?

On account of the importance of the flue gases, Table V is here printed. This table shows the analysis obtained theoret-

TABLE V

FLUE GASES

Analysis.			By Weight.		
Gas	Test	Check		Wet	Dry
CO_2	16.082	16.730	Oxygen	21.967	23.249
CO	0.135	0.140	Carbon	4.200	4.444
O_2	11.476	7.930	Nitrogen	68.330	72.307
N_2	72.307	75.200	Moisture	5.514	

Pounds dry gas.....10,004
Pounds H_2O..... 584

In all the calculations it has been necessary to consider the moisture present in the materials used and the gases formed. The heat taken out by the moisture, superheated to the high temperaures involved amounts to a large figure, and is an important part of all the calculations. It will be noted in the analysis of the raw materials charged that soft ore runs as high as 20.26 per cent moisture. Economy would be obtained by drying such material before charging.

TABLE VI

OPEN HEARTH. SUMMARY SHEET MATERIALS

Material	Pounds	Si	SiO_2	Mn	MnO	P	P_2O_5	Fe	Fe_2O_3	Fe_3O_4	C	Free O_2	Absorbed O_2	S
Charged	3,330,900	21,027	.....	36,079	.....	6,251	.....	3,153,997	.....	40,195	72,076	11,089	42,595	1,290
Addition No. 1	805,550	.....	26,404	.....	2,136	.....	578	.....	327,623	17,032	28,485	75,965	25,322	.....
Run off slag	109,084	.....	10,930	.....	10,067	.....	2,180	.....	.....	.....	.....	104,808	.....	.....
Normal slag run	260,379	.....	40,358	.....	20,600	.....	7,408	.....	.....	.....	.....	.....	.....	.....
Normal slag to the ladle	130,189	.....	20,179	.....	10,300	.....	3,704	.....	.....	.....	.....	.....	.....	.....
Furnace steel	3,345,087	.....	.....	6,000	.....	700	.....	3,334,097	.....	.....	3,000	2,645	.....	1,290
Recarburizer	285,376	3,438	241	14,780	.....	828	.....	252,165	.....	.....	12,613	.....	4,396	487
Product	3,618,490	724	.....	17,910	.....	861	.....	3,586,000	.....	.....	14,040	.....	.....	1,548
Ladle slag	142,894	.....	26,237	.....	13,992	.....	.....	.....	.....	.....	.....	1,695	.....	.....
To ladle gases	1,929	.....	.....	.....	.....	.....	.....	.....	.....	.....	1,573	59	.....	229
To furnace gases	291,719	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Material	FeO	Al_2O_3	CaO	$CaFl_2$	MgO	Moisture	
Charged	.....	.....	.....	.....	.....	.....	NOTE
Addition No. 1	.....	10,386	201,329	4,227	43,810	67,568	To Gases
Run off slag	39,000	3,279	37,070	.....	6,558	.....	C..... = 97,561
Normal slag run	50,117	4,738	109,506	2,818	24,835	.....	O_2..... = 126,590
Normal slag to the ladle	25,058	2,369	54,753	1,409	12,417	.....	H_2O... = 67,568
Furnace steel	.....	.....	.....	.....	.....	.....	.....
Recarburizer	.....	1,084	68	.....	.....	.....	.....
Product	.....	.....	.....	.....	.....	.....	.....
Ladle slag	25,393	3,453	54,821	1,409	12,417	.....	.....
To ladle gases	.....	.....	.....	.....	.....	.....	.....
To furnace gases	.....	.....	.....	.....	.....	.....	.....

TABLE VII

MATERIALS PER TON OF INGOTS. TEST OF FURNACE NOS. 89 AND 90.

Material	Pounds	In the Furnace	Pounds
Hot metal	1,120.00	Carbon oxidized	42.750
Cold pig	96.30	Silicon oxidized	13.015
		Manganese oxidized	18.620
Total	1,216.30	Phosphorus oxidized	3.436
		Iron oxidized to FeO	70.660
Blooms	590.30		
Plate	19.50	Fe_2O_3 reduced	202.800
Ingot butts	59.17		
Rails	39.30	Fe_3O_4 reduced	35.420
Tin scrap	26.61		
Turnings	3.44	H_2O liberated	41.820
Steel scrap	103.90		
Stirring rods	2.84	$CaCO_3$ decomposed	64.660
Total steel scrap	844.06		
		In the Ladle	Pounds
Fine ore	200.90		
Cinder	5.63		
Scale	5.50		
Dolomite	68.70		
Limestone	153.80	Silicon oxidized	1.679
Lump ore	61.90	Carbon oxidized	.975
Fluor spar	2.75	Manganese oxidized	1.776
		Phosphorus oxidized	.412
Total	499.18	Aluminum oxidized	.254
		Sulphur oxidized	.141
Run off slag	67.50		
Two-thirds normal slag	161.15		
ADDITIONS:			
Hot metal	164.75		
Ferro manganese	8.60		
Ferro silicon	.65		
Sulphur	.226		
Coal	2.135		
Aluminum	.254		
Total additions	176.615		
Furnace metal	2,070.00		
Product	2,240.00		
Difference	170.00		
In recarburizer	176.615		
To ladle slag	7.865		
One-third normal slag	80.575		
Total ladle slag	88.440		
Dry producer gas used	2,510.00		
Water (downtake)	72.00		
Water (valve)	96.80		
Dry air used	7,378.00		
Water in air	106.00		
Dry flue gas	10,004.00		
Water in flue gas	584.00		

Table VII gives the pounds of materials used in the production of one ton of ingots, together with the quantities of slags and gases produced.

In the figuring of the heat balance, which follows, the figuring is based on heat per ton of ingots.

The above table would also be of value in open-hearth furnace design, when considering the gases used and formed.

Table VI gives in a condensed form the arrangement of the various materials in pounds, giving the amounts of material charged, slags formed, and steel produced, and from which table an adjustment of the oxygen was obtained. That is the amount of oxygen freed, and the amount of oxygen absorbed in the various reactions, together with the oxygen coming into the furnace from the gas mixture, gives the amount of free oxygen that enters into the burned gases passing out of the furnace.

All data taken for the figuring of the heat balances are operating figures such as are found in practice, since test conditions are often in error when taken as representing average practice.

The following heat balance naturally divides itself up into many parts, and for this reason on the left-hand side of each sheet appears the data and results in obtaining the heat involved at each stage of the process, while on the right-hand side of the sheet is printed the calculations involved in producing the result.

In the calculations effecting the heat balance the methods employed are those of Prof. J. W. Richards, as given in his most valuable work, "Metallurgical Calculations." Figures used in obtaining the mean specific heat of the materials involved, and symbolized SM, are those constants given by Dr. Richards. These constants for the different materials and elements are, in nearly all cases, figures for the specific heat at 0.0 deg. C., or 32 deg. F.; hence for obtaining the specific heat at a higher temperature the figure at 32 deg. F. is increased by the addition to this figure of the product of a known constant and the temperature range. The results are given in detail in the calculations.

In figuring the latent heat in gases due to the moisture content, the metric system, as given by Dr. Richards, is used, and the result obtained in kg. cal. converted into B.t.u.

In collecting the data it was very difficult to obtain any reliable information in relation to the moisture contents of air and gas, but the figures as here given were checked a number of times, and accordingly can be taken as representative of the conditions met with in the regenerator cellars in open-hearth plants.

It will be noted that the hot gas efficiency of the producers' figures out to be 94.99 per cent, and that the loss of heat after passing the reversing valves amounts to 3.22 per cent, so that the net efficiency of the producers delivering gas to the furnace regenerators is 91.77 per cent. The gas flues on the furnaces tested were underground, and the question naturally comes up: Would there be as great a loss of heat if the flues were carried overhead and not buried as in this case?

Producer Gas

Calculation of the Total Heat in the Producer Gas Before Being Burned

HEAT CHART AND DISTRIBUTION. GAS UP TO FURNACE PORTS

(A) HEAT TO PRODUCERS. PER TON OF INGOTS

Material	Lbs.	B.T.U.-Lb.	B.T.U.	T°	Range °	SM (O-t) x-	B.T.U., Sensible
Coal	510.0	13,427.2	6,847,872	70°	38°	0.33522	6,497.4
Steam	149.0	1,170.7	174,434	...			
Air (dry)	1,995.5	.....	.....	90°	58°	0.23421	27,107.2
H_2O in air	28.5	1,104.49	31,478	90°	16°	0.42597	193.8
				Sp. 74°			
H_2O in coal	5.6	.....	.....	70°	38°	1.00000	212.8

Total latent B.T.U. ..... = 7,053,784 Total sensible B.T.U. = 34,011.2
Total B.T.U. in materials ..... = 7,087,795

(B) PRODUCER GAS MADE AND IN DOWNTAKE FROM PRODUCER

Material	Lbs.	B.T.U.-Lb.	B.T.U.	T°	Range °	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1400°	1368°	0.30130	1,034,371
H_2O in gas	72.0	1,109.07	79,853	89°SP	1279°	0.56090	51,640

Total latent B.T.U. ..... = 5,647,033 Total sensible B.T.U. = 1,086,017
Total B.T.U. in producer gas ..... = 6,733,050
Hot gas efficiency of producer ..... = 94.99%

(C) PRODUCER GAS AT ENTRANCE TO REGENERATOR AND AFTER PASSING WATER COOLED VALVE

Material	Lbs.	B.T.U.-Lb.	B.T.U.	T°	Range °	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1100°	1,068°	0.29488	779,640
H_2O in gas	96.8	1,112.13	107,654	99°SP	979°	0.53000	50,239

Total latent B.T.U. ..... = 5,674,834 Total sensible B.T.U. = 829,879
Total B.T.U. in producer gas delivered to entrance of regenerator .. = 6,504,713
Efficiency of producer making gas delivered to furnace ..... = 91.77%
B.T.U. lost in cooling up to regenerator entrance 228,337
= 3.22%

(D) GAS AFTER PASSING THROUGH GAS REGENERATOR. TEMPERATURE OF REGENERATOR BEING 2,120°F.

Material	Lbs.	B.T.U.-Lb.	B.T.U.	T°	Range °	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1,940°	1,908°	0.312774	1,497,780
H_2O in gas	96.8	1,112.13	107,654	99°SP	1,908°	0.616520	113,837

Total latent heat B.T.U. 5,674,834 Total sensible B.T.U. 1,611,617
Total B.T.U. in the gas through regenerators ..... = 7,286,451
Heat in B.T.U., gained in regenerator ..... = 781,738

CALCULATIONS

(A)

Coal. B.T.U. = 14,544 C.×62,000 (H_2-O_2+8)+3,996 S.
= 11,086.9 ×2,294.0+46.3 = 13,427.2 B.T.U.

Steam. Forty-five-pound gauge. Sixty-pound absolute.

Coal. SM. (32°−70°), temperature difference = 38°.

C	76.23%	0.1567+38×0.00036	=0.17038	0.12988
H	5.02%	3.3700+38×0.00017	=3.37646	0.16949
N	1.38%	0.2405+38×0.0000119	=0.24054	0.00332
O	10.61%	0.2104+38×0.000010	=0.21079	0.02236
Ash	5.65%	0.1800	0.18000	0.01017
SM. Coal				0.33522

Air. SM. (32°−90°), temperature difference = 58°.

N	76.9%	0.2405+58×0.0000119	=0.24119	0.19547
O	23.1%	0.2104+58×0.0000104	=0.21100	0.04874
SM. Air				0.23421

Moisture in air = 9 grains per cubic foot. Saturation point = 74°F.
74°F. = 23.33°C. 606.5+0.305 (23.33) = 613.61 kg. cal.
= 1,104.49 B.T.U. latent.
Superheat = 90°−71° = 16°F. SM. = 0.42 + 0.000103 t°F.,
= 0.42597
Sensible heat = 0.42597×28.5×16° = 193.8 B.T.U.

(B)

Producer gas. SM. (0°−t°). Temperature range = 1,368°. T. = 1,400.

CO_2	10.317%	0.1900 +1,368×0.00006	=0.27208	0.028070
CO	25.003%	0.2405 +1,368×0.0000119	=0.25678	0.064192
O_2	.025%	0.2104 +1,368×0.0000104	=0.22462	0.000056
CH_4	1.711%	0.5300 +1,368×0.000150	=0.73520	0.012579
C_2H_4	.458%	0.3670 +1,368×0.000130	=0.54484	0.002495
H_2	1.000%	3.3700 +1,368×0.000170	=3.60256	0.036025
N_2	61.486%	0.2405 +1,368×0.0000119	=0.25678	0.157883
SM. Producer gas				0.301300

Gas weight = 0.071103 pounds per cubic foot. 14.06 cubic feet per pound.
Moisture = 14.3 grains per cubic foot. 14.3 grains H_2O = S. P. of 89°F.
89°F. = 31.65°C. 606.5+0.305 (31.65) = 616.15 Kg.-Cal. = 1,109.07 B.T.U.
SM. (0°−t°) = 0.42+0.000103 t°. t° = 1,368°. S.M. = 0.56090.

(C)

Temperature range = 1,068°F.

CO_2	=	0.25408	0.026220
CO		0.25321	0.063302
O_2		0.22150	0.000055
CH_4		0.69020	0.011806
C_2H_4		0.50584	0.002326
H_2		3.55150	0.035515
N_2		0.25321	0.155664
SM. gas			0.294888

Moisture per cubic foot gas = 19.1 grains. Saturated at 99°F.
99°F. = 37.2°C. 606.5+0.305 (37.22) = 617.85 Kg.-Cal. = 1,112.13 B.T.U
SM. (0°−t°) = 0.42+0.000103 t° = 0.53000.
Superheat = 1,068−99 = 979°F.

(D)

Temperature range = 1,908°F.

CO_2	0.30448	0.031361
CO	0.26320	0.065800
O_2	0.23024	0.000057
CH_4	0.81620	0.013965
C_2H_4	0.61504	0.002816
H_2	3.69436	0.036943
N_2	0.26320	0.161832
SM. gas		0.312774

Moisture in gas = 19.1 grains. SP. = 99°F.
SM. = 0.42+0.000103 t° = 0.61652.

Air

Calculation of the Total Heat in the Air for the Combustion of the Producer Gas

AIR. HEAT CHART AND DISTRIBUTION
HEAT IN AIR BEFORE PASSING THROUGH REGENERATOR

a. (A)

Material	Lbs.	T_1	T_2	Range	SM (0°−t)	Latent B.T.U.	Sensible B.T.U.
Dry air	7,378	90°	32°	58°	0.23421	.....	100,340
H_2O in air	106	SP. = 74°	Superheat = 16°			117,797	.....
Total B.T.U. in air ..... = 218,137							

(B)

Material	Lbs.	T_1	T_2	Range	SM (0−t)	B.T.U. Sensible	Latent B.T.U.
Dry air	7,378	90°	2,012°	1,922°	0.256422	3,636,573	.....
H_2O in air	106	...	.....	.....	0.627236	130,906	117,076

Total B.T.U. after regenerator ..... = 3,884,555 B.T.U.
Heat absorbed by the air in regenertaor ..... = 3,884,555
Heat originally in the air ..... = 218,137
Absorbed ..... = 3,666,418

(C)

Heat in unburned gas at ports ..... 7,286,451 B.T.U.
Heat in regenerated air at ports ..... 3,884,555 B.T.U.
Heat to furnace ..... 11,171,006 B.T.U.

CALCULATIONS

(A) AIR BEFORE REGENERATOR

SM (32°−90°).

N	76.9%	0.2405+58×0.0000119	=0.24119	0.19547
O	23.1%	0.2104+58×0.0000104	=0.21100	0.04874
SM. air				0.23421

Water in air = 9 grains. Latent heat = 1,104.49×106 .. = 117,076 B.T.U.
Superheat = 16°F. SM. = 0.42597.
Sensible heat = 0.42597×106×16 ..... = 721 B.T.U.

S.M. AIR

(B) TEMPERATURE OF AIR OUT OF REGENERATOR = 2,012°F.

SM. (32°−2,012°).

N	76.9%	0.2405+1 980×0.0000119	=0.264062	0.203063
O	23.1%	0.2104+1,980×0.0000104	=0.230992	0.053359
SM. air				0.256422

Moisture in air, 9 grains per cubic foot. Total weight, 106 pounds.
SP. = 74°F. Latent B.T.U. ..... = 1,104.49 B.T.U.
Superheat = 2,012−74° ..... = 1,938°
SM. = 0.42+0.000103 t°F ..... = 0.627236
Latent heat in water vapor = 106×1,104.49. ..... = 117,076
Sensible heat = 106×0.627236×1,938 ..... = 130,906

Materials

OPEN HEARTH. SUMMARY

a. (A) Heat Distribution. Heat in Materials Charged

Material	Pounds	T°	SM (0−t)	SM × (T−32°) × Lbs.	B.T.U.-Lb.	Gross B.T.U.
Hot metal	1,120.00	2050	.....	.....	441	493,920
Cold pig	96.30	70	0.12095	442.98	.....	443
Blooms	590.30	70	.....	.....	.....	.....
Plate	19.50	70	.....	.....	.....	.....
Butts	59.17	70	.....	.....	.....	.....
Rails	39.30	70	.....	.....	.....	.....
Tin	26.61	70	.....	.....	.....	.....
Turnings	3.44	70	.....	.....	.....	.....
Steel	103.90	70	.....	.....	.....	.....
Rods	2.84	70	.....	.....	.....	.....
Total steel	844.06	70	0.11650	3,714	.....	3,714
Fine ore	200.90	70	0.14959	.....	.....	1,145
Cinder	5.63	70	0.14869	.....	.....	63
Scale	5.50	70	0.14869	.....	.....	.....
Solomite	68.70	100	0.21790	.....	.....	1,016
Limestone	153.80	100	0.20860	.....	.....	2,184
Lump ore	61.90	100	0.14959	.....	.....	619
Fluor spar	2.75	100	0.21540	.....	.....	40

Cold material. Total B.T.U. 503,144

OPEN HEARTH. SUMMARY

Calculations

a. (A)
Cold pig. SM (0°−t°) = SM (32°−70°) = 0.1200 + 0.000025 t.
= 0.12095
96.30 × 38 × 0.12095 = 442.98 B.T.U.
Steel = 844.06 × 38 × 0.11650 = 3,714 B.T.U.
Lump ore = 0.1456 + 0.000105 t = 0.14959 = SM.
Cinder scale = 0.1447 + 0.000105 t = 0.14869 = SM.

(B) Heat Generated by Oxidation of Materials

Material	Pounds	Product	Base B.T.U.-Lb.	Gross B.T.U. Generated
Carbon oxidized	42.750	CO_2	14,580	623,295
Silicon oxidized	13.015	SiO_2	14,094	183,433
Manganese oxidized	18.620	MnO	3,114	57,982
Phosphorus oxidized	3.436	P_2O_5	10,620	36,490
Iron oxidized	70.660	FeO	2,430	171,704

B.T.U. generated by oxidation 1,072,904

Total Heat to Furnace

In gases	7,286,451 B.T.U.
In air for combustion	3,884,555 B.T.U.
In materials charged	503,144 B.T.U.
By oxidation in furnace	1,072,904 B.T.U.
Total B.T.U. to furnace	12,747,054 B.T.U.

Heat Absorbed

Heat Absorbed in the Furnace
Heat Absorbed in Reduction, Etc. (A) Pounds Per Ton of Ingots

Material	Pounds	B.T.U.-Pound	Gross B.T.U.
Fe_2O_3 reduced	202.80	2,270.0	460,356
Fe_3O_4 reduced	35.42	2,137.0	75,692
H_2O liberated	41.82	2,678.8	112,027
CO_2 (lime) liberated	153.80	666.0	102,431

Total B.T.U. absorbed ..... 750,506 B.T.U.

Heat Absorbed in Slag Formation. (B)

Slag	Pounds	B.T.U.-Pound	Gross B.T.U.
Run off slag	67.50	1,134	76,545
Two-thirds total normal slag	161.15	1,226	197,570
One-third total normal slag to ladle slag	80.57	1,226	98,779
Ladle slag	88.435	(*)	

Total B.T.U. in slags run from furnace ..... = 372,894 B.T.U.
*Not in furnace, hence not calculated.

Heat in the Melted Steel. (C).

Material	Pounds	T°F.	Total B.T.U.-Lb.	Gross B.T.U.
Dead melted steel	2,070.0	3,012°	622.8	1,289,196

Heat Utilized in Furnace

Heat of reduction, etc.	750,506 B.T.U.
Heat in run off slag	76,545 B.T.U.
Heat in normal slag	296,349 B.T.U.
Heat in melted steel	1,289,196 B.T.U.
Total B.T.U. utilized	2,412,596 B.T.U.

In the foregoing calculations it will be noted that the gas producers were taken as a separate unit in the amount of heat produced, and which likewise follows for the air required for the combustion of this gas. The heat in the cold materials as charged into the furnace is also considered since the normal temperature of this material is considerably above 32 deg. Fahrenheit. The heat due to oxidation of the carbon, silicon, and other ingredients in the metal by the iron ore charged is also treated separately, as it may be noted in the calculation.

The summary of the above calculations is as per the following table, which gives the total heat which is delivered to, or produced within the furnace, and the sum total of which gives the first part of the heat balance. The distribution and utilization of this heat is as per the calculations which follow:

(A) Calculations

Water evaporation. 70°−212°. 606.5 + 0.305 (67.5°) = 627.08 Kg.-C.
= 1,128.74 B.T.U.
SM. (O−t) = 0.42 + 0.000103 t. t = 2,500° ..... S M = 0.67750
B.T.U., superheat = 1 × 0.67750 × 2,500° − 212° ..... = 1,550.1
Latent ..... = 1,128.7
Total ..... = 2,678.8 B.T.U.-Lb.

(B) Slags
Run off slag. MP. = 1,350°F.
t° = 1,350.
MS. = 0.16359 (1 + 0.00039 t).
SM. = 0.16359 (1.5265)
SM. = 0.24972 at MP.

Run Off Slag

Analysis	%	Sp. H. 0°	SM.
SiO_2	10.02	0.1833	0.01836
Al_2O_3	3.00	0.2081	0.00624
FeO	35.75	0.1460	0.05219
MnO	9.23	0.1511	0.01394
CaO	33.98	0.1715	0.05827
MgO	6.03	0.2420	0.01459
			.16359

Latent heat fusion = (1,350 + 273) (0.24972) = 405.3 Kg.-C. × .225 = 91.2 Kg.-Cal.
A balance of the heats for formation = 200 Kg.-Cal.
Total heat in slag up to MP. = 337 + 91 + 200 = 628 Kg.-Cal.
= 1,134 B.T.U.-Lb.
t° = 1,450.
SM. = 0.16970 (1 + 0.00039 t°).
= 0.16970 (1.5655)
= 0.26566
Latent heat of fusion.
(1450 + 273) (0.26566) (0.225) = 103 Kg.-Cal.
Lf. = 103 Kg.-Cal.
H. = 385 Kg.-Cal.
Hf. = 150 Kg.-Cal.
Total.. = 638 Kg.-Cal.

Normal Slag

Analysis	%	Sp. H. 0°C.	SM.
SiO_2	15.50	0.1833	0.02841
MnO	7.91	0.1511	0.01195
FeO	19.25	0.1460	0.02810
Al_2O_3	1.82	0.2081	0.00378
CaO	42.05	0.1715	0.07211
MgO	9.54	0.2420	0.02308
CaF_2	1.08	0.2100	0.00227
			0.16970

Normal Slag. MP. = 1,450°
Superheated to 1,600°C. = 150° × 0.27559 ..... = 41 Kg.-Cal
Total B.T.U. per pound of slag 1,22

Heat in Steel. (C). SM. = 0.19887 − (23.44 ÷ t°) = 0.18427
Heat in melted steel at 1,350°C. = 300 Kg.-Cal. 250° × (0.18427) = 46 Kg.-Cal.
346 Kg.-Cal. = 622.8 B.T.U.

Gases

Calculation of the Heat Carried Out of the Furnace in the Gases

Having obtained the distribution of heat for the various reactions occurring in the furnace proper, it becomes necessary to consider the flue gases. Table V gives the analysis of the flue gases obtained from the elementary distribution as compared to actual analysis. The oxygen distribution is obtained under "A," following.

OPEN HEARTH. SUMMARY. (CALCULATIONS)

(A) FLUE GASES:

Ultimate analysis. CO_2 = 16.73 C = 4.56 O_2 = 12.17
CO = 0.14 C = 0.06 O_2 = 0.08
= 4.62 = 12.25

CO_2. C = 4.56 = 98.70% = 4.386. CO_2 = 16.082
CO. C = 0.06 = 1.30% = 0.058. CO = 0.135

CO_2. O_2 = 11.696 O_2 from all gases .... = 23.249
CO. O_2 = 0.077 O_2 above ..... = 11.773
11.773 Free O_2 ..... = 11.476

(B) LATENT HEAT LOSS IN FLUE GASES:

CO. = 0.135. CP. = 3.49 B.T.U.-Cu. Ft. 0.135 × 3.49 = .47 B.T.U.-Cu. Ft.
0.47 × 114,751 = 53,923 B.T.U.

Cubic feet of flue gases made:

	Sp.wt.-32°		
CO_2	16.082	0.12424 = 0.01998	Dry gas made = 10,004 pounds.
CO	0.135	0.07857 = 0.00011	10004 ÷ 0.08718 = 114,751 cubic feet.
O_2	11.476	0.08953 = 0.01027	H_2O = 584 pounds.
N_2	72.307	0.07859 = 0.05682	= 35.6 grains-Cubic Feet.

Weight-Cubic Foot flue gas 32° ..... 0.08718 pounds

(C) MIXTURE OF GASES BEFORE COMBUSTION

Gas	Pounds Dry (X)	Pounds Wet	T° Regenerator End (Y)	SM.	SM × X × Y
Producer	2,510	2,582	1,940°	.31277	1,523,002
Air	7,378	7,484	2,012°	.25642	3,806,435
Flue	10,004	10,588	.....	....	.....

Producer X × Y = 4,869,400
Air ..... = 14,844,536
19,713,936 ÷ 10,004 = t° ..... = 1,993°F.
Temperature of gas and air mixture before combustion ..... = 1,993°F.
Temperature of bath at end of heat ..... = 3,012°F.
Temperature of regenerators ..... = 2,190°F.

(D) ELEVATION OF TEMPERATURE.

			T° UNKNOWN
CO_2	16,082	0.19000 + 0.00066	= 0.03055 + 0.00000964
CO	.135	0.2405 + 0.0000119	= 0.00032 + 0.00000001
O_2	11.476	0.2104 + 0.00001040	= 0.02414 + 0.00000119
N_2	72.307	0.2405 + 0.0000119	= 0.17390 + 0.00000860

T = 0.22891 + 0.00001944 T

Water at 35.6 grains S.P. = 120°F. = 49°C.
Then 606.5 + 0.305 (49) = 621.4 Kg.-C. = 1,118.7 B.T.U.
SM. = 0.42 + 0.000103 (T° − 120°).

$$\text{Elevation } T° = \frac{\text{B.T.U. generated}}{\text{Weight gases} \times \text{specific heats}}$$

B.T.U. generated by gases and in furnace.
Sensible heat in gases (producer) ..... = 1,611,617
Latent B.T.U. in producer gas ..... = 5,674,834 B.T.U.
B.T.U. by oxidation, etc ..... = 1,072,904 B.T.U.
Total ..... = 6,747,738 B.T.U.
Less loss of CO value ..... = 53,933 B.T.U.
B.T.U. generated in combustion ..... = 6,693,805 B.T.U.

$$T° = \frac{6{,}693{,}805 \text{ B.T.U.}}{10{,}004 \times (0.22891 + 0.0000194\,T) + 584 \times (0.42 + 0.000103\,T)}$$

T = 6,693,805 ÷ (2,290 + 0.194077 T + 245 + 0.060152 T).
Then 2,535 T + 0.254 T^2 = 6,693,805. Quandatic. $ax^2 + bx + c. = D$.

$$x = \frac{-b \pm \sqrt{b^2 - ac}}{a}$$

Then T° = 2,160°F. = Elevation of temperature.
Temperature of air and gas mixture ..... = 1,993°F.
Elevation of temperature ..... = 2,160°F.
Temperature combustion ..... = 4,153°F.

In order to obtain the heat deposited in the regenerators it is necessary to know the temperature at which the gases leave the furnace. In order to obtain this it is necessary to obtain the temperature of the gas mixture, and this is accomplished in calculation (C). The elevation of temperature is then obtained by calculation (D), which gives the temperature of combustion. The heat absorbed in the regenerators is then obtained as in calculation (E) and (F). Finally the heat carried to and up the stack is obtained as in calculation (G).

Calculation (D) gives the temperature of combustion. There has been considerable discussion as to just what flame temperature is reached in the open hearth. Since the material in the furnace reaches a temperature of over 3000° F., it would naturally follow that the temperature of combustion is considerably higher than the furnace itself. This high temperature of 4153° F., as the temperature of combustion, would argue that a longer furnace might be more economical, and give greater heat absorption by the materials in the furnace, and probably less heat loss up the stack.

(E) HEAT ABSORBED IN REGENERATORS. TEMPERATURE OF GAS.

Temperature of the burned gases at regenerator end = 2,832°F. (Observed).
Pounds of dry gas = 10,004. Sensible heat = SM. = 0.22891 + (0.00001944 t°)
Moisture ..... = 584. = 0.22891 + 0.055054
= 0.28396

T° (Range) = (2,832 − 32°) = 2,800°.
Sensible heat = 2,800 × 0.28396 × 10,004 ..... = 7,954,080 B.T.U.
Water vapor = S.P. = 120°F. Latent heat at 120° ..... = 1,118.7 B.T.U.
SM. = .42 + 0.000103 (2,832 − 120) = 0.42 + 0.000103 (2,712°).
SM. = 0.699336.
Superheat in B.T.U. = 0.699336 × 1 × 2,712 ..... = 1,896.6 B.T.U.
Latent heat above ..... = 1,118.7 B.T.U.
Total heat ..... = 3,015.3 B.T.U.
per pound of moisture
Heat in the moisture = 584 × 3,015.3 ..... = 1,760,935 B.T.U.
Sensible heat in gases ..... 7,954,080 B.T.U.
Total passing to regenerator ..... 9,715,015 B.T.U.
Latent heat in CO unburned ..... 53,933 B.T.U.
Total heat ..... 9,768,948 B.T.U.

(F) GASES PASSING OUT OF THE REGENERATORS TO THE STACK.

Temperature of the gases as observed ..... 1,252°F
Sensible heat. SM. = 0.22891 + (0.00001944 t°).
= 0.22891 + (0.00001944) (1,252)
= 0.22891 + 0.025116 ..... = 0.25324
T° (range). = 1,252 − 32° = 1,220°. Sensible heat = 10,004 × 0.25324 × 1,220.
= 3,090,748 B.T.U.
Water vapor. S.P. = 120°F. Latent heat = 1,118.7 B.T.U.
Superheat = 1,252 − 120° = 1,132°F. SM. = 0.42 + 0.000103 (1,132°).
= 0.53660
Superheat in B.T.U. = 0.53660 × 1,132 × 1 = 607.4 B.T.U. per pound moisture
Latent heat ..... = 1,118.7 B.T.U.
Total heat in moisture vapor ..... = 1,726.1 B.T.U. per pound
Total heat in water vapor = 584 × 1,726.1 ..... = 1,008,042 B.T.U.
Heat in gases per above ..... = 3,090,748 B.T.U.
Heat from unburned CO gases ..... = 53,933 B.T.U.
Heat out of regenerators. Total ..... 4,152,723 B.T.U.
Heat deposited in the regenerators. In ..... = 9,768,949 B.T.U.
Out ..... = 4,152,723 B.T.U.
Heat deposited in regenerators ..... = 5,616,225 B.T.U.

(G) GASES PASSING TO THE STACK.

Heat in gases passing out of regenerators ..... = 4,152,723 B.T.U.
Average temperature of the stack as observed ..... = 1,200°F.
SM. = .22891 + 0.00001944 × T° ..... = 0.24161
Sensible heat in gas = 10,004 × 0.24161 × (1,200 − 32°) .. = 2,823,056 B.T.U.
Water vapor = S.P. = 120° ..... } = 1,118.7 B.T.U.
Latent heat ..... }
SM. = 0.42 + 0.000103 (1,200 − 120°) ..... = 0.53124
Superheat B.T.U. = 0.53124 × 1 × 1,080 ..... = 573.7 B.T.U.
Latent heat ..... = 1,118.7 B.T.U.
Total ..... = 1,692.4 B.T.U.
per pound
Total heat in water vapor = 584 × 1,692.4 ..... = 988,361 B.T.U.
Heat in gases ..... = 2,823,056 B.T.U.
Total heat ..... = 3,811,417 B.T.U.
Heat in unburned CO ..... = 53,933 B.T.U.
Total heat up the stack ..... = 3,865,350 B.T.U.
Total heat delivered to the regenerators ..... = 9,768,948 B.T.U.
Total heat going out of the regenerators ..... = 4,152,723 B.T.U.
Heat deposited ..... = 5,616,225 B.T.U.
Heat gained in regenerators by producer gas ..... = 781,738 B.T.U.
Heat gained in regenerators by air ..... = 3,666,418 B.T.U.
Heat absorbed by gases ..... = 4,448,156 B.T.U.
Heat deposited ..... = 5,616,225 B.T.U.
Heat absorbed ..... = 4,448,156 B.T.U.
Heat lost ..... = 1,168,069 B.T.U.
Efficiency of regenerators ..... = 79.22%
Heat in gases passing to flue to stack from regenerators = 4,152,723 B.T.U.
Heat going up the stack ..... = 3,865,350 B.T.U.
Heat loss in flue ..... = 287,373 B.T.U.

Table VIII gives the summary of the heat distribution and is the heat balance.

Conclusion

HEAT BALANCE.

Heat Distribution Per Ton of Ingots

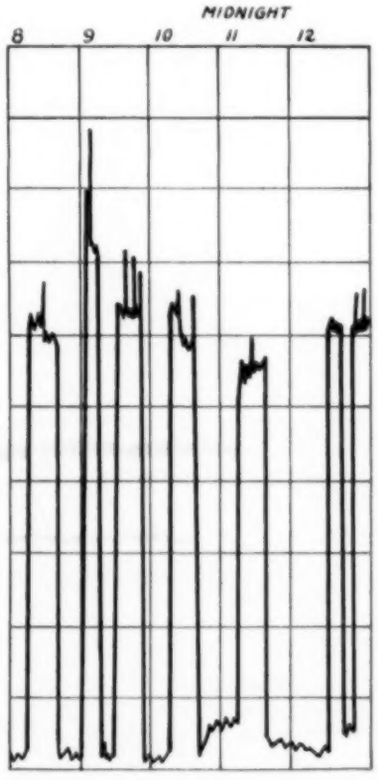
	B.T.U.	%
Heat in materials charged to producers	7,087,795	.....
Heat in producer gas in downtake	6,733,050	.....
Efficiency of producer (hot gas)	.....	94.99
Producer gas at entrance to regenerators	6,504,713	.....
Efficiency of producer	.....	91.77
Heat in producer gas after passing through the regenerators	7,286,451	.....
Heat gained in the regenerators	781,738	.....
Heat in air before passing to regenerators	218,137	.....
Heat in air before passing through regenerators	3,884,555	.....
Heat gained by air in regenerators	3,666,418	.....
Heat in gas and air going to furnace	11,171,006	.....
Heat in materials charged into furnace	503,144	.....
Heat generated by oxidation in furnace	1,072,904	.....
Total heat delivered to furnace	12,747,054	100.00
Heat absorbed in reduction	750,506	5.88
Heat absorbed in slag formation	372,894	2.92
Heat absorbed in melting steel	1,289,196	10.11
Total heat used in useful work	2,412,596	18.91
Heat in gases passing to regenerators	9,715,015	76.21
Heat loss in CO gas unburned	53,933	.42
Heat loss by radiation, etc., furnace proper	565,510	4.46
Heat in gases passing out of regenerators	4,152,723	32.57
Heat deposited in regenerators	5,616,225	44.06
Heat taken out by air and gas in regeneration	4,448,156	34.89
Efficiency of regenerators	.....	79.22
Heat loss in flue to stack	287,373	2.25
Heat going up the stack	3,865,350	30.32
Total heat delivered or generated in furnace	12,747,054	100.00
Useful work	2,412,596	18.91
Radiation at furnace	565,510	4.46
Heating of regenerators	5,616,225	44.06
Radiation in flue	287,373	2.25
Loss up stack	3,865,350	30.32
Total	12,747,054	100.00

From the above heat balance it will be noted that the amount of heat actually used in the melting of the steel amounts to but 10.11 per cent of the heat in the fuel, and the total amount of heat used in performing useful work is 18.91 per cent, that is, the efficiency of an open-hearth furnace, giving the furnace credit for reduction, slag formation, and melting metal, is less than 20 per cent. There are a number of ways of utilizing the waste heat from the furnace, one of which is the utilization of waste heat boilers connected to the stack. Aside from this, and considering the furnace as a heating unit, there is great room for improvement in the furnace itself.

In modern boiler houses and engine rooms there are gauges, etc., giving information as to the working of the apparatus, and in many plants there are steam engineers that make regular use of the indicator as a guide to engine performance, but the use of gauges, etc., in the open hearth, has unfortunately not met with a favorable reception. Some plants have meters on their gas lines to each furnace, but this is the exception and not the rule, and where the meters are applied they are on natural-gas furnaces.

The value of having a recording mechanism on a furnace may be noted from the following. On one of the furnaces from which this heat balance was figured, there were 509 reversals for the week. The average time for each reversal was 10 minutes, the longest period on one set of regenerators was 70 minutes, and the shortest period was 2 minutes. In open-hearth practice it is the custom for each turn to keep up one end of a furnace. That is the night shift might have the left end of a furnace and the day shift might have the right end of a furnace to keep up. One turn naturally favors its end of the furnace, and in the operation of the furnace will do all that it can to favor its side and keep it from cutting away, thereby placing all the wear and tear on the other end, as the case may be.

This practice leads to irregularity in the operation of the regenerators, and soon there is a good and a bad end to the furnace. The adjoining illustration shows the difference in time involved in the reversals. A long period and then a short period on a regenerator naturally does not give a proper distribution of heat to the regenerators, and one set of regenerators will be at a lower temperature than the other. At certain periods in an open-hearth heat it is necessary to have quick reversals, but there is no reason why these reversals



should not and could not be made of equal duration for each set of regenerators, thereby maintaining them at a balanced temperature.

If it were the practice to have all furnaces equipped with a recording mechanism which would show the reversals, the superintendent would be aware of what took place at all hours, and, by paying more attention to the utilization of heat in a furnace, would have a very good opportunity to effect economy.

The above heat balance was for a set of two furnaces of 60 tons capacity, with a hearth 32′ 0″ x 14′ 9″, such a 60-ton furnace showing an efficiency of 18.91 per cent.

There is a wide discrepancy in open-hearth design, as may be noted by the following.

Plant.	Capacity.	Hearth.
Duquesne	60 Ton.	32′ 0″ x 14′ 9″
Ohio Works	60 Ton.	29′ 0″ x 14′ 11″
Gary	60 Ton.	40′ 0″ x 15′ 3″
Republic	60 Ton.	32′ 0″ x 15′ 0″
Lackawana	60 Ton.	43′ 0″ x 16′ 9″

There is an even greater variation in the dimensions of the regenerators, flues, etc., in the above furnaces than there is in the hearth, hence it would follow that the utilization of heat would be different, and it would be profitable to all steel manufacturers to know just what design would be the most effective for the result desired. The logical way to determine this is from a metallurgical standpoint, which in this case would be in heating efficiency.

Pittsburgh, Pa.

Rational Absorption of Hydrochloric Acid*

By Dr. Theodor Meyer

Whoever tries to acquaint himself with the methods and apparatus for the production of muriatic acid from hydrochloric acid gas and reads the chapter of more than 100 pages in Lunge's great treatise may easily have the sensation of wheels turning in his head, as said by the scholar in Goethe's "Faust." Surely, I do not want to depreciate Lunge's excellent work as the completeness and critical exactness of his exposition of the subject can hardly be surpassed. But the processes and apparatus discussed by him are of such a variety, the propositions and views even partly contradictory and often irrational, that much acumen and care is necessary to discover what is suitable and rational in the chaos.

As Lunge shows, even Hurter's writings and experiments based on sound science and on practical experience contain many errors and lead in part to wrong conclusions such as might not be expected from an expert like Hurter. A point in question is, for instance, the representation of the coke tower as the most excellent condensation apparatus. The acme of absurdity is the "invention" of Schloesing, who finds precooling of the gases to be detrimental and concludes that to

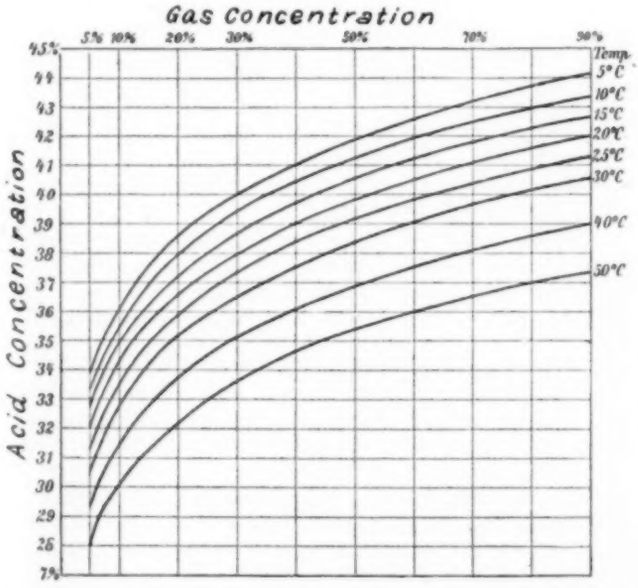


FIG. 1.—ABSORPTION CURVES FOR DIFFERENT GAS CONCENTRATIONS AND TEMPERATURES

aid the absorption of the gases their temperature and vapor pressure should be as high as possible.

It is well known that the absorption of hydrochloric acid is an exothermic reaction. According to Berthelot when x grams of HCl are dissolved in a gram of water $477.5x - 157x^3$ calories are produced; for instance, the heat evolved in the formation of hydrocloric acid of 20° Baume corresponds to a rise of temperature of 118° C. (212° Fahr.). It is therefore evident that the absorption will be easier and quicker the better means are provided to carry off this newly produced heat.

But the maximum concentration of the hydrochloric acid produced also depends on the temperature. For instance, when the hydrochloric acid is absorbed from the gas of 5% concentration at 20° C. (68° Fahr.) it is still possible to produce acid of standard concentration (32% HCl), while at a temperature of 50° C. (122° Fahr.) a higher concentration than 28% cannot be obtained. On the other hand, at 5° C. (42° Fahr.) a concentration of almost 34% can be obtained. According to Hurter's calculations, at a temperature of t° C. one gram of water is able to absorb 0.3040—0.0016 t gram HCl.

The second most important factor which determines the absorption of hydrochloric acid is the content of HCl in the gas. apparatus for the production of muriatic acid from hydrochloric acid gas and reads the chapter of more than 100 pages in the gas mixture.

These two laws are the essential basis of the rational absorption of hydrochloric acid. They are illustrated by the curves in Fig. 1 which have been plotted according to the formula of Hurter:

$$C = (0.3040 - 0.0016\,t)\,P^{0.125}$$

in which C is the grams of HCl absorbed by 1 gram of water at the temperature t and P is the partial pressure of the HCl gas in millimeters of mercury.

The most important rule which may be deduced from these curves for the requirements of practice is to keep the concentration of HCl in the gas mixture at not less than 20 or 30% if possible.

Besides the above considerations a number of other points must also be considered, especially the influence of the diluting air on the heat phenomena and the content of water in the gas mixture; the effect of cooling on the water precipitation by which latent heat is again set free; the speed of the gas; the impurities in the gas; the relation of hydrochloric acid of different concentration on the excess of air, and finally the particular properties of the materials used for the apparatus.

Above all it is clear that a good and quick absorption of the HCl can only be accomplished by most intimate contact of the gas with water. This contact may be provided in general by three methods which are different in principle.

(1) The gas is subdivided into very small bubbles and is passed through the water by pressure or suction.

(2) The water is subdivided into exceedingly small droplets and made to act on the gas.

(3) As large a liquid surface as possible is provided to be acted on by the gas.

The method (1) is useful in the laboratory, but is not suitable for manufacturing on a large scale, since if the depth of the liquid is low the gas bubbles pass through it too quickly, so that there is not sufficient time for an intimate contact. On the other hand, if the depth of the liquid is great a rather considerable amount of work would be required for the gas to overcome the pressure of the liquid.

The method (2) was discussed unfavorably by Hurter in 1893 (*Journal* Society Chemical Industry, page 226). His criticisms were justly refuted by Lunge, though they could be understood since the patented process of Newall and Bowman of 1874 had not proven successful. But two points must be taken into consideration in explaining this lack of success. Firstly, at that time no direct means had yet been evolved to subdivide liquids into such a very fine spray as is possible to-day. Secondly. Newall and Bowman tried to apply the method in a wrong manner. It is well known that muriatic acid with more than 25% HCl loses HCl when in contact with air at a temperature of 15° C. (59° Fahr.) and this loss of HCl is, of course, the quicker the more intimate the contact with the air. As the temperature increases that strength of acid which will remain constant in the presence of air is still further reduced. For instance, at 60° C. (140° Fahr.) it is 23% HCl. This shows that muriatic acid of at least 32% (20° Baume) could be obtained by means of a water spray only if the HCl concentration in the gas is correspondingly high.

But, even in this case, trouble is caused by the heat of absorption which manifests itself the more, the less diluent gases are present which could take it up. These two points were not sufficiently taken into consideration by Newall and Bowman. As they carried out the process the spray of water acted directly on the gases coming from the furnace and this reaction took place in sandstone vessels which were connected by means of stoneware pipes, so that there was little opportunity for the escape of the heat evolved. We will show later that the use of spray water can give very good results if it is applied in the correct way and at the correct place.

The method (3) is the one which is being generally and almost exclusively used at present, although details of operation

*Translated from *Zeitschrift für Angewandte Chemie*, February 21, 1913. p. 97.

vary greatly and apparatus of greatly different forms are employed, such as towers with coke filling or stoneware filling, Lunge plate towers, stoneware vessels, and stoneware tourills, and Cellarius cooling and absorption vessels.

In England, the mother country of the manufacture of hydrochloric acid, the big coke towers in connection with some stoneware vessels have remained chiefly in use. On the other hand, in continental Europe stoneware tourills with a final washing tower of stoneware of moderate dimensions are preeminent. The Schaffner (Aussig) system with towers of 8 to 12 meters (26 to 40 feet) height filled with stoneware dishes are in particular favor.

The large coke towers, which have a height up to 36 meters (120 feet), are generally built of sandstone, so that there is little chance for the air to produce a cooling action. If the uncooled gases enter such a tower at the bottom a moderate cooling effect is obtained by the evaporation of the muriatic acid trickling down from above so that the heat becomes latent and as such is returned to the higher portions of the tower where condensation takes place and the latent heat becomes free again, while the condensed acid trickles down again, evaporates partly anew and so on.

The net effect of what happens is, therefore, that the heat is transferred to a somewhat higher point, while the real condensation process is restricted to the highest and coolest part.

The hydrochloric acid formed in the highest and coolest part of the tower must, therefore, pass through the lower hot portion, and as the constant concentration of muriatic acid at the boiling point is not more than 20% or 13° Baume, it is evident that it is very difficult to get in this way a fairly high concentration of the acid.

Moreover, in many cases the tower is not even able to reduce the concentration of HCl in the exit gases to the desired degree, so that it becomes necessary to pass the gas finally through a washing tower and the low acid obtained from it is a source of trouble.

Formerly the hydrochloric acid was used principally for the Weldon process, especially in England, and for this purpose a concentration of 16 to 18° Baume was sufficient. At present the conditions have been changed, especially in continental Europe, where the Weldon process has been replaced almost completely by the electrolytic process. To have a commercial value, hydrochloric acid should have at present a concentration of not less than 20° Baume, and for some purposes, like the manufacture of aniline salts, a considerably higher concentration, up to 24° Baume, is required.

The system of a number of tourills with a final washing tower, which is used largely in Germany, is undoubtedly superior to the large coke tower, since part of the sensible heat and of the heat of absorption is carried off in advance by the tourills with their relatively large outside surfaces and since the rather thin stoneware walls of the tower carry off far more heat than the sandstone walls in the British system, which are six times thicker.

Nevertheless, even in the German system about one-half of the absorption still takes place in the towers, especially if the inaccessible surface of the liquid in the tourills becomes covered with fatty impurities. This trouble occurs often when some oil or grease is put into the saltcake pan to prevent ebullition. In such cases a failure of the tourills often occurs and it may happen that in order to get the acid up to 20° Baume it must flow off at 15 to 17° Baume from the tower and that the tower becomes so hot as to crack the walls. In a system designed in a rational manner with pre-cooling and cleaning of the gases the acid flowing off from the tower has a concentration of only 1 to 2° Baume.

Effective pre-cooling cannot be obtained with air, especially with chemical stoneware as the only material that is really to be taken into consideration a wall-thickness of 10 to 15 millimeters (0.4 to 0.6 inch) must be used. The specific heat of water is four times that of atmospheric air if figured per weight, and more than 3,000 times that of atmospheric air if figured per volume.

After Fryer and others had made unsuccessful experiments, water-cooling was successfully introduced in connection with the invention of the absorption vessel of Lehmann and Cellarius in 1899, although their absorption vessel was originally not intended for water-cooling and could not be used with water-cooling (*Angew. Chem.*, Vol. 21, page 1069, 1908). When in 1901 I had to solve the problem to get hydrochloric acid of 24° Baume from the gas of the Oehler muffle furnace, I tried to employ the new Cellarius absorption vessel by modifying its construction for water-cooling. The success was excellent.

The gas coming from the furnace, after having been cooled to air temperature by passing it through 11 empty water-cooled vessels, was passed through a system of 35 absorption vessels placed in water circulating in the opposite direction. In spite of pre-cooling and absorption cooling, a rise of temperature up to 80° C. was still obtained. This occurred approximately in the twelfth vessel and proves the intensity of the absorption. The acid running off from the front vessel could easily be obtained with a concentration 23.5 to 24 or even 24.5° Baume. This concentration changed, of course with the fluctuations in the concentration of the gas, which amounted to about 40%.

If the maximum temperature is displaced toward the exit, this should be taken as a warning to restrict the water supply somewhat. It is thus possible to control the operation of the absorption process with this type of apparatus by means of a few thermometers, which is much simpler than the measurement of the specific gravity, although the latter may also be observed in a convenient manner.

After having been passed through the water-cooled vessels the acid was passed through a few air-cooled vessels and then through a washing tower, which yielded an acid of 1.5° Baume when supplied with pure water.

In 1902 C. Uebel devised a vessel similar to the Cellarius tourill which is, however, not intended for circulation of liquid through it, but especially for cooling of gases and vapors, for instance, in nitric acid manufacture. In order to distinguish it from the Cellarius absorption tourill it might be properly called a condensation tourill. It is being used now generally for pre-cooling the hydrochloric acid gases which it robs of their sensible heat and latent heat, while the Cellarius absorption tourill takes away the heat of absorption while it is being formed.

In the course of time several more or less different forms of vessels have been designed, which, however, have no advantages over the original patented construction. Yet it cannot be claimed that the Cellarius tourill already satisfies all requirements.

In an ideal absorption vessel the gas and the liquid should act on each other in counter-current. Since the absorption makes the liquid heavier the liquid should enter at the top and leave the vessel at the bottom. In the Cellarius vessel, however, the liquid enters and leaves at the bottom. Further, the liquid should enter where the gas goes out and the liquid should flow off where the gas enters. In the Cellarius vessel, however, the liquid enters and leaves alternately at the entrance and exit of the gas. The liquid, therefore, circulates in one-half of its vertical as well as horizontal passageways only in counter-current with the gas, and in the other half flows in the same direction as the gas.

Moreover, an effective counter-current arrangement is rendered difficult by the shortness of the passageway of the gases, which, measured between the centers of the gas inlet and the gas outlet, is only 50 centimeters (19 inches) for size I and 70 centimeters (28 inches) for size H.

This short length of the Cellarius vessels has another disadvantage. The front side of the vessels should be accessible by means of a passageway, say 1 meter (3.3 feet) wide, while the width of two rows of tourills with their ends placed against each other is 225 centimeters (7½ feet), including cooling boxes for size II, so that almost one-third of the floor space of the building is required for the passageways.

Finally, it cannot be denied that the form of the Cellarius

tourill is a little complicated, or at least that a simpler form could surely be made at less cost.

The design shown in Fig. 2 is intended to realize the advantages of the Cellarius tourill while avoiding its disadvantages. The gas passes here in the ordinary manner from the back to the front of the absorption vessel, then from the front to the back in the next absorption vessel, and so on. The passage way of the gas in each vessel is 140 centimeters (55 inches). The gas and the liquid are always in counter-current, since the liquid also enters the successive absorption vessels alternately at the front and at the back. The liquid always enters near the surface and leaves at the bottom. As these absorption vessels are 160 centimeters (63 inches) long, hardly one-fourth of the floor space of the building is required for the passageways. On account of their simple form the manufacture of these absorption vessels involves no difficulties.

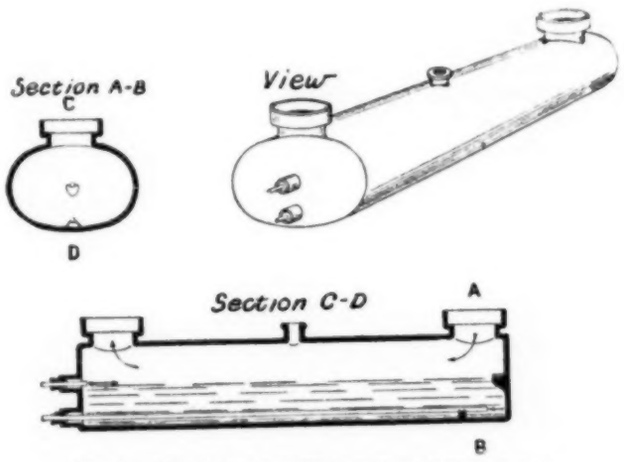


FIG. 2.—ABSORPTION VESSEL FOR HYDROCHLORIC ACID

An advantage of the Cellarius vessel is the quiet surface, which is due to the fact that the liquid enters and leaves below the surface. The minimum height of the level of the liquid is the height of the saddle. Since my new absorption vessel has no saddle, it might seem that the level of the liquid could not be maintained properly. However, it is easy to see that the level depends entirely upon the drop from the exit from one vessel to the influx into the next vessel, or in case of the last vessel, upon the inclination of the exit pipe. If the vessels are properly arranged with a descent of 10 to 15 millimeters (25/64 to 19/32 inch) from one vessel to the next vessel, it is easy to adjust the level of the liquid so that it is a few millimeters above the influx.

In one respect the quiet surface of the liquid is liable to become a source of trouble. If the gas is not free from particles insoluble in water and if it contains especially fatty particles, these are liable to form a film on the surface of the liquid so as to prevent absorption. When this happens it is easily observed by the fact that the vessel is cool. Even when using pure materials this was found to happen, and the cause of the trouble was discovered to be a film of a white substance like paraffine, which seemed to have had its origin in the rock salt. By means of a gas filter such troubles can be avoided with certainty.

Instead of two gas inlet and outlet pipes three or four may be easily attached to each absorption vessel, if this is desirable for the distribution or the mixing of gases. If the vessel is intended to serve for condensation only, the only change necessary is to omit or to close the upper pipe for the entrance of the liquid. The same vessel may, therefore, be used for absorption as well as condensation.

While an extended quiet surface of the liquid is most suitable for the production of a strong acid it is by no means suitable for extracting the acid from dilute gases, which becomes necessary at the finish of every condensation process. At that point it is proper to use a finely sub-divided spray of water or dilute acid whereby a most intimate mutual contact can be obtained without danger of reducing the efficiency by the excess of air or by the heat of absorption, which is hardly noticeable in this case.

In the following I give comparative data for both types of absorption vessels, the figures of the Cellarius tourill referring to size II.

Cellarius Tourill II.	*New Absorption Vessel.*
Length 103 cm., or 40 in.	160 cm., or 63 in.
Width 66 cm., or 26 in.	55 cm., or 22 in.
Total Volume	
110 liters, or 29 gallons.	173 liters, or 45.7 gallons.
Liquid Volume	
40 liters, or 10.6 gallons.	88 liters, or 23 gallons.
Gas Volume	
70 liters, or 18.5 gallons.	85 liters, or 22.5 gallons.
Absorption Surface	
0.55 sq. m., or 6 sq. ft.	0.88 sq. m., or 9.5 sq. ft.
Cooling Surface	
1.4 sq. m., or 15 sq. ft.	2.3 sq. m., or 25 sq. ft.

Since only above 24 to 25% muriatic acid loses in concentration by contact with air it might appear practical at the first glance to carry out the absorption process by means of a spray of water up to that limit and to absorb only the balance by means of a quiet surface of water. But this would be a mistake. Firstly, because the absorption apparatus for spray liquid cannot be easily constructed for water-cooling and, secondly, because the spraying requires power, which means an extra expense.

So far the absorption of the last traces of acid from the gas has been always carried out in washing towers as far as possible. Undoubtedly, in a tower there is a much more intimate contact between the gas and liquid than in vats or tourills, but not nearly as intimate a contact as in spray apparatus.

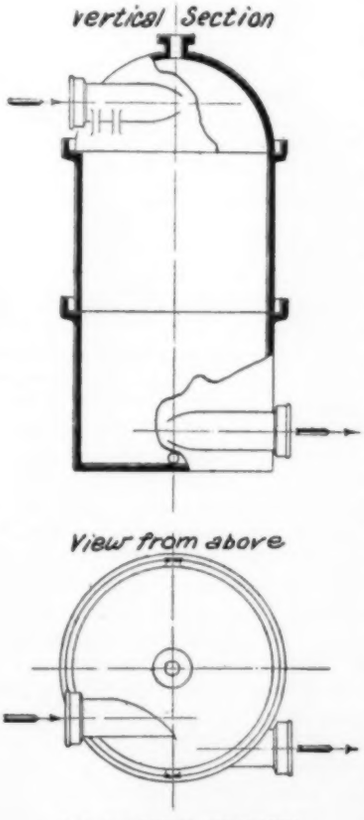


FIG. 3.—SPRAY APPARATUS

Experiments made on a large scale have given, for instance, the following figures for the concentration per cubic meter: before the spray cylinder 56 grams HCl, 1.9 grams SO_3, 3.5 grams SO_2, and behind the spray cylinder 2.2 HCl and 2.8 grams ($SO_3 + SO_2$) figured as SO_4.

The final washing tower is in a sort of an intermediate position and has, therefore, no right of existence whatever for the absorption of hydrochloric acid. Where the object is to get the last traces of acid from the gases and to enrich the dilute acid thereby obtained up to a concentration of 4 to 6° Baume the most preferable procedure is to use water spray process and to circulate the obtained dilute acid directly through the absorption vessels in order to bring it up to the concentration which can be obtained with the acid concentration prevailing in the gas.

In conjunction with the Deutsche Ton- und Steinzeugwerke

in Charlottenberg* I have designed an apparatus for the treatment of dilute hydrochloric acid gas with spray water.

The principles of this apparatus for which a patent has been applied is based on the following:

(a) The gas and the spray of water are to be brought into as intimate contact as possible so that the gas is "hydratized," if we may use this expression.

(b) There should be no "dead corners" where there is no reaction, nor should the gas have any opportunity to avoid in part the action of the spray of water.

(c) The gas should be brought in a very slow and uniform current in contact with the spray of water.

(d) The spray of water should be in excess.

For this purpose the pipes through which the gas enters and leaves the apparatus are arranged in a tangential direction, the one at the conical or hemispherical top of the apparatus, the other at the bottom, while the spray nozzle yielding or emitting the water spray is best placed in the center of the top. (Fig. 3.)

The capacity of the nozzle is chosen in such a way that it gives almost the whole, but a little less than the whole quantity of the water which is necessary for recovering the HCl from the furnace in form of muriatic acid of the desired concentration. The hydrometer or thermometer readings determine the exact adjustment of the operation, which is carried out by supplying the small additional necessary amount of water directly into the last absorption vessel. The regulation of the acid concentration is, therefore, very much simpler than with the present concentration system with the final tower where it is necessary to ascend the tower in each case.

Especially with high coke towers this is very inconvenient. It should be pointed out that it is not sufficient as Lunge writes (Handbuch, Vol. II, third edition, page 375) to control the concentration of the finished acid several times during the day, for when this is found too low it is then too late to repair matters since the whole apparatus is filled with acid of too low a concentration. The daily measurement and adjustment of the supply of water to the tower is therefore absolutely necessary. This is, of course, also necessary whenever there is any change in the amount or number of the charges.

In the final tower (excepting, perhaps, Lunge's plate tower) about one-half of the volume is occupied by the filling material. For this reason and on account of its low efficiency it is necessary to make the tower high, several times higher than the spray apparatus. The lifting of the water to this height entails an expense which is practically as much as the expense for generating the pressure in the spray apparatus.

For towers of chemical stoneware the width is limited for constructive reasons, the maximum being about 1.4 meters (4.6 feet). Since about one-half of the cross-section in the same is taken up by the filling material, we have, with the spray apparatus of the same width, double the effective cross-section, while the velocity of the gas is reduced to one-half. However, the greatest advantage of the spray apparatus is the simplicity and cheapness of construction as compared with the final tower with its heavy foundation and high frame work and stairways.

According to the size of the hydrochloric acid plant and other circumstances one or more spray apparatus may be used and these may be connected in series or in parallel.

*The European Associates of the German American Stoneware Works.

It is advantageous to place behind the spray apparatus one or more gas filters, filled with granulated acid-resisting material, in order to catch the acid particles contained in the gases as mist (*Angew. Chem.*, Vol. 19, page 1313, 1906).

By using this new spray apparatus it is possible to recover the acid in a much more complete manner and to clean the exit gases in a much better way than heretofore. The acid is completely removed from the gas so that it is possible to let the gas pass directly into the atmosphere. It is not necessary to pass the gas into the smokestack, thereby greatly impairing the draft, so that it is necessary to build it higher.

The other rules which are to be taken into consideration for the absorption of hydrochloric acid, as well as constructive details, have been described by me elsewhere (Th. Meyer, Sulfat und Salzsäure, Halle, 1907).

I will, therefore, discuss only a few points bearing on the present subject.

The principle is essentially as follows: The more or less wet and hot gases, having been brought to a concentration as high as practicable, pass from the furnace with the least possible artificial suction into the pre-cooling apparatus. This consists of a preliminary empty vessel and a series of condensation tourills.

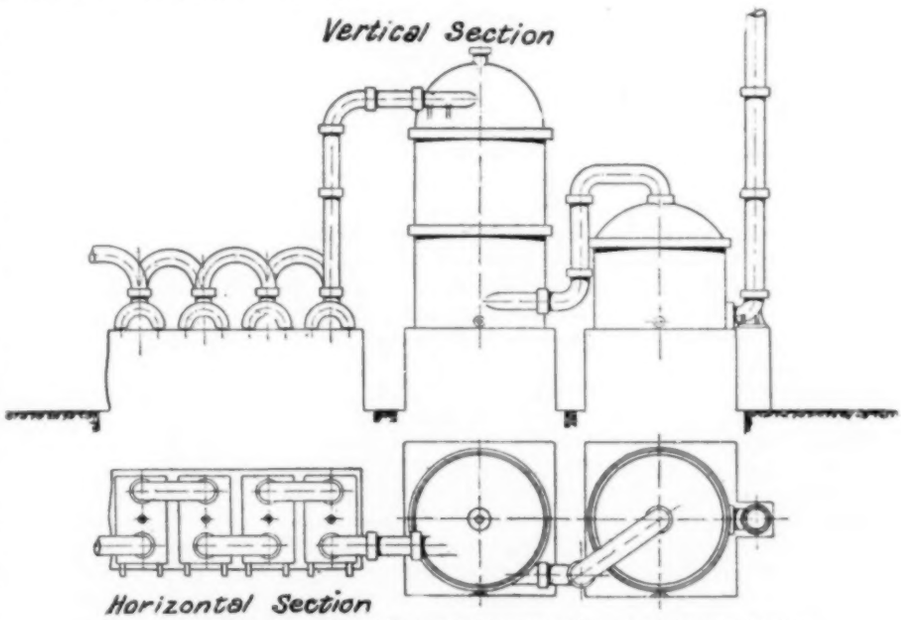


FIG. 4.—VERTICAL AND HORIZONTAL SECTIONS OF FINAL PART OF APPARATUS

At the beginning of this system a little water is added if necessary. The gases then pass through a stoneware exhauster and immediately afterwards through a gas filter; after having thus been purified and cooled, the gases enter into the series of water-cooled Cellarius tourills or absorption vessels of my approved design described above. (Fig 3.)

They finally pass through one or more water spray apparatus and if necessary through a final gas filter. Fig. 4 shows the horizontal and vertical section of the final part of the apparatus.

With respect to the concentration of the hydrochloric acid gas, the apparatus for the decomposition of the sodium chloride by sulphuric acid is of chief importance. The reverberatory furnace which yields the most dilute gas is rarely found now at least in Germany, but in the same class there are the gas from the Hargreaves process and the gas from the decomposition of magnesium chloride which is carried out on a commercial scale in the Stassfurt district, where the condensation of the HCl vapors is indeed very difficult.

The gas from the muffle is also very low in concentration. According to Lunge (Handbuch, Vol. II, 3 Aufl., pages 276 and 277) the average concentration was 1.3% HCl in one case and 2.6% in another case in the Chemische Fabrik Griesheim. According to the curves in Fig. 1 it is impossible

to get a 32% acid from such a gas, even if a very low temperature is employed such as is impractical on a commercial scale.

If the much richer gas from the pan which contains from 10 to 30% HCl would be mixed with the gas from the muffle a gas would be obtained from which acid of normal concentration could be obtained (although the concentration of the mixture would not be the arithmetical mean, since the volume of gas from the muffle is eight times that from the pan).

Usually, however, this mixing is undesirable, because pan acid and muffle acid are of different purity.

There is, however, another very simple way which is apparently little known; to bring the muffle acid up to 20 or 20° Baume. It is simply necessary to place the first one to the three tourills carrying the muffle acid in front of the absorption battery on the pan side. The pan acid is, of course, not connected into these tourills. I do not know if this easy way is used in factories.

It is certainly preferable to operate in advance with concentrated gases. I have worked on this problem for many years. The result has been on one side the Oehler furnace for bisulphate and on the other side the mechanical muriatic acid retort (German patents 186,398 and 238,570).

The mechanical retort requires sulphuric acid of approximately 90%, but the product of absorption is in proportion less impure.

With the Oehler furnace, consisting of several connected muffles, it is possible to get a gas containing 40 to 50% HCl in the average. With the retort it is possible to get an acid of 60 to nearly 100%.

If the retort is placed behind the Oehler furnace so that the firing of the Oehler furnace also serves for the retort and if the bisulphate coming from the Oehler furnace is further treated in the retort this combination represents a complete substitute for the ordinary saltcake furnace and besides other advantages it is able to yield a HCl gas of high concentration which can be easily condensed.

The pipes which conduct the gas from the furnace are liable to break pretty often, whether they are made of fire-clay, stoneware, glass or quartz. In order to avoid this it is advisable to insulate them well up to the point where they enter the pre-cooling apparatus, if it is not preferred to make them of sandstone.

Directly next to the preliminary vessel there is the battery of pre-coolers. In this combined system, as well as in the exhauster and the gas filter, so much impure acid is condensed as corresponds to the humidity and the temperature of the gas.

With the saltcake pan, the Oehler furnace, and also with the retort, the condensed acid amounts to 3 to 6% of the total acid. From the much more dilute and drier muffle gas not enough acid would be condensed; moreover, the muffle gas would enter the pre-cooling system at too high a temperature. For this reason a little water is added through the nozzle into the preliminary vessel, which must have for this purpose an inside height of 1.5 to 2.5 meters (5 to 8 feet). This water spray has a cooling action. Part of the heat is rendered latent and the water vapor thus formed is taken along to the center of the cooling system where it is condensed together with sulphuric acid and other impurities.

If the cooling system is arranged with cooling water running uniformly in counter-current so that it leaves the first vessel at a temperature of 80 to 100° C. (176 to 212° Fahr.) the gas may enter at a temperature of 200° C. without any danger of breaking.

The gas filter was first used in hydrochloric acid manufacture by Boulouvard (French patent 284,139, 1899), but at the wrong place, since it was placed directly after the furnace. But the gas filter can be efficient only when acting on the cooled gas. The particular object of the gas filter is to precipitate particles carried along in a misty form. Together with these acid particles the greatest part of the impurities is precipitated, especially sulphuric acid, iron chloride, etc., as well as some of the arsenic. But the filter also serves to catch any fatty particles which would prevent the absorption.

One of the chief requirements of proper operation of the apparatus is the dimensioning of the pipes and of the draft. Stoneware pipes cannot be built like lead or iron pipes for any diameter. Hence the rule not to make the units of a hydrochloric acid plant too large.

Further, when designing a new plant, care must be taken to calculate as exactly as possible the quanities of gas which will be evolved per unit of time and to calculate from this the speed of the gas per second. The cooling action and condensation, as well as the absorption, depend, of course, under otherwise identical conditions, on the time required for working off the charge.

There are many sins committed against this principle, especially with ordinary saltcake plants, in so far as tourills and towers of the same type and size are used for the absorption system of both pan and muffle and only the number of tourills connected in series (instead of in parallel) is sometimes somewhat larger for the muffle. In the illustrations of Lunge, for instance, figures 138 and 152, both the pipes for the pan gas and the muffle gas are exactly the same kind and, therefore, of the same cross-section.

As an example, let us take a large saltcake furnace in which 5,300 kilograms of sodium chloride are treated in 24 hours producing about 1,930 cubic meters HCl at 0° C., distributed as follows:

From the pan 1,160 cubic meters HCl = 5,800 cubic meters gas of 20% concentration at 0° C. = 8,560 cubic meters gas at 130° C.

From the muffle 770 cubic meters HCl = 38,590 cubic meters gas of 2% concentration at 0° C. = 62,580 cubic meters gas at 170° C.

Therefore the rate per second is as follows:

Pipe line from pan: 99 liters gas after the furnace = 67 liters after the pre-cooling system.

Muffle pipe line: 724 liters gas after the furnace = 447 liters after the pre-cooling system. If an internal diameter of 25 centimeters (9⅞ inches) is assumed for the pipes carrying the cooled gas, we find a speed of flow of the gas of 1.37 meters per second in the pipes for the pan gas and a speed of 14.77 meters per second in the pipes for the muffle gas.

Even if, for the muffle gas, the pipes of 25 centimeters diameter should be replaced by pipes of 45 centimeters (18 inches) diameter (the largest which are available) the speed of the muffle gas would still be 4.55 meters per second, that is, three times the speed of the gas from the pan.

Pipes with a diameter of 45 centimeters (18 inches), however, can hardly be used between the Cellarius tourills, even if the largest size is chosen, since the gas inlets and outlets have a diameter of only 30 centimeters (11¾ inches).

There is, therefore, nothing left but to split up the absorption system for the muffle gas into two or more lines in parallel. For instance, for the construction of the gas filter, as made by the Deutsche Ton- und Steinzeugwerke according to my design, I always specify one filter for the pan and three in parallel for the muffle gas.

Even the best condensation apparatus cannot be maintained absolutely tight. Always a little air will enter and the more air will enter in the greater the suction. For this reason it is very important to maintain the draft not higher than is necessary.

The new spray apparatus can also be used in combination with the exhauster and gas filter advantageously for the auxiliary condensing system, the object of which is to avoid the escape of acid vapor during the operation when the furnace doors are open.

The dilute hydrochloric acid obtained in this way (for instance, in an Oehler furnace 3.7° Baume, 4.8% HCl, 0.24% H_2SO_4, and 0.007% Fe_2O_3), may be used even though not especially pure, as an addition to the main absorbing system so as to increase the total output of muriatic acid.

The great problem of the muriatic acid industry—the

mechanical furnace—is still a problem of the future and for the present there is little hope that it will soon be solved. But, even so, I trust I have shown that there are still many promising problems to be solved with respect to increasing the efficiency of operation in mechanical as well as hygienic respects.

Offenbach on Main, Germany.

An Electrolytic Theory of the Corrosion of Iron.*

By Bertram Lambert

The application of electrolytic ideas to the question of the corrosion of iron has been discussed by Whitney,[1] Cushman,[2] Tilden,[3] Walker,[4] Armstrong[5], and many others, but much confusion and misunderstanding has arisen owing to the different interpretations used by the various writers.

The American writers have discussed the question from the standpoint of Nernst's conception of the source of electromotive force between a metal and a solution. It seems to follow from their reasoning that all iron (even if it be chemically pure and physically homogeneous) ought (a) to rust in contact with air (or oxygen), (b) to cause the deposition of copper from solutions of copper salts.

The present writer's experiments[6] have proved beyond any doubt that iron can be prepared which will not rust on long exposure to air and water (either distilled or ordinary tap-water), and will not cause the deposition when placed in even strong solutions of copper sulphate or copper nitrate under ordinary conditions.

It also seems to be thought by many supporters and opponents of electrolytic theories that any electrolytic theory of the corrosion of iron must "stand or fall" on the question of the solubility of iron in pure air-free water.

The object of this paper is to show that a simple and natural development of the ideas of Faraday on electrolysis will give us the beginnings of a satisfactory theory of the corrosion of iron—a theory incomplete as yet owing to the lack of experimental facts, but one which is quite in accordance with well-established facts and which is not affected by the question whether iron is soluble to any appreciable extent in pure air-free water.

It is well known that while ordinary commercial impure zinc readily dissolves when placed in dilute sulphuric acid, the pure metal is scarcely affected unless metallic connection is made between it and another more electronegative metal, either within or without the liquid. Under these conditions the pure zinc dissolves, but the action ceases immediately the metallic contact between the pure zinc and the more electronegative element is broken.

In impure zinc the more electronegative element is already present within the mass of the metal, in the form of minute impurities, such as copper or lead, or perhaps as another form of the metal which is relatively electronegative to the main mass of the metal.

Whenever we have two metals (or two modifications of the same metal) which are electrically different, that is, have different solution pressures, and they are placed in metallic contact in an electrolyte, then the relatively electropositive part will dissolve with the production of an electric current flowing through the electrolyte from the electropositive to the electronegative pole, and in the possite direction through the metal.

The rate of such a reaction depends: (*a*) on the magnitude of the difference of electric potential—that is, the difference of solution-pressure between the two metals or the two modifications of the same metal, and (*b*) on the resistance offered by the electrolyte to the passage of the electric current.

No part of the metal can dissolve unless an electric current actually passes through the electrolyte. The rate of the reaction may, however, be so infinitesimal that the amount of metal passing into solution will not be sufficient to respond to chemical tests even after long periods.

Let us consider, in this light, the case of a piece of ordinary commercial iron in contact with the electrolyte water.

Such a piece of iron is impure and not homogeneous—there are some parts of it which have a different solution-pressure from other parts, and so when it is placed in contact with the electrolyte water we have all the conditions for the production of an electric current.

If the conditions are such that an appreciable electric current can pass between the two electrically different parts of the iron the metal will dissolve at the relatively electropositive parts. The fact that iron is practically insoluble in pure water (in the absence of oxygen) shows that the current which actually does pass is so infinitesimal that the amount of iron dissolved cannot be detected, even after long periods, by chemical means.

This may be due to two causes, namely, (*a*) the small magnitude of the electromotive force owing to the small differences of potential between the electrically different parts of the metal, and (*b*) the great resistance offered to the passage of an electric current by the electrolyte.

The writer's experiments[7] seem to be generally accepted as proving beyond any doubt that commercial forms of iron will undergo corrosion quite readily in contact with pure water and pure oxygen in the complete absence of carbonic acid or any other acid—that the only essentials for the corrosion of ordinary iron are water and oxygen. It is generally believed that iron must pass through a process of solution before rust is produced, and so, while the metal is practically insoluble in pure water alone, it must be soluble in the presence of oxygen. It follows, then, that oxygen must bring about some alteration in the conditions of the "voltaic circle"—commercial iron and water—in such a way that a greatly increased electric current passes between the electrically different parts of the iron.

That oxygen can bring about such a change of conditions is beautifully illustrated by an experiment described by Dunstan and Hill.[8] Cushman[9] has shown that if a piece of commercial iron is left in contact with a solution of sodium chloride to which a little phenolphthalein has been added, a pink color develops in the solution which is in contact with some parts of the metal. These color zones, which are always confined to a portion (or portions) only of the metal, indicate the presence of hydroxyl ions (or free alkali) at these points. After being kept some time, the metal beneath the pink zones remains quite bright, but corrosion takes place gradually at the other parts of the metallic surface.

This experiment shows that the solution is being electrolyzed, and that the iron is passing into solution at those parts not marked out by the pink color of the phenolphthalein. It shows also that some parts of the iron are electrically different from other parts, and that the electromotive force produced by these differences of solution-pressure is enough, under these conditions, to cause the passage of an electric current of sufficient magnitude to produce perceptible electrolysis of the solution of sodium chloride.

It was shown by Dunstan and Hill, however, that the extent of this electrolysis was conditioned by the presence of oxygen, because, in the absence of air (or oxygen), the pink zones were not developed, and therefore an electric current sufficient to produce perceptible electrolysis could not pass.

Oxygen, therefore, must do one of two things—it must in some way or other increase the electrical differences between the parts of the metal and so increase the electromotive force,

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*A paper read before the Faraday Society at the Manchester Meeting, Friday, April 4, 1913.

[1]Journ. Amer. Chem. Soc., 1903, 25, 394.
[2]Trans. Amer. Elektrochem. Soc., 1907, 12, 403.
[3]Trans. Chem. Soc., 1908, 93, 1356.
[4]Journ. Iron and Steel Inst., 1909, 1, 70.
[5]Science Progress, 1911, 22, 311.
[6]Trans. Chem. Soc., 1912, 101, 2068 et seq.
[7]*Loc. cit.*, pp. 2056 *et seq.*
[8]*Trans. Chem. Soc.*, 1911, 99, 1853.
[9]Amer. Soc. Testing Materials, 1907.

or it must reduce the resistance of the circuit. When a piece of commercial iron is put into water in a vacuum, iron strives to pass into solution at the relatively electropositive part of the metallic surface, but hydrogen, produced by the electrolysis of the water, is deposited at the relatively electronegative part. This film of hydrogen, forming almost simultaneously and covering up the negative pole, introduces an enormous resistance into the circuit, and reduces the electric current to an almost negligible strength, so that the rate at which the iron passes into solution is infinitely small.

Oxygen dissolved in the water probably acts by deoxidizing the hydrogen deposited at the negative pole—destroying the polarization—and so allowing a greater current to pass between the electrically different parts of the metal.

If this argument is true, then commercial iron ought to pass into solution, in the absence of oxygen, if it is placed in an electrolyte, such as copper sulphate solution, where, instead of the non-conducting hydrogen film, there would be a conducting film of metallic copper produced at the relatively electronegative pole. Experiment shows this to be true. Commercial iron, when brought into contact with a solution of pure copper sulphate in a vacuum, causes the immediate deposition of copper on the iron just as readily as when the experiment is conducted in the presence of air; in a short time all the copper is removed from the solution and iron takes its place.

It seems to be clear then that when commercial iron rusts in contact with air and water, oxygen plays an important part in the actual dissolving of the iron, quite separate and distinct from its later function of converting the dissolved iron into rust.

The formation of rust from the dissolved iron is not, as it is sometimes supposed, a simple oxidation of the ferrous ions. It is not possible for the ferrous ion to be oxidized to the ferric ion by means of ordinary molecular oxygen. This can readily be proved by a simple experiment. If a solution of ferrous sulphate is made strongly acid with sulphuric acid and oxygen is passed through the solution, comparatively little ferric salt will be found in the solution even after the oxygen has been passed through it for several days. The greater the proportion of free sulphuric acid present, the less the oxidation which takes place. The oxygen only oxidizes the undissociated ferrous hydroxide formed by the hydrolysis of the ferrous salt, and since ferrous hydroxide is a very weak base, hydrolysis is very considerable unless a large excess of acid is present.

In the rusting of iron, then, the metal passes into solution as ferrous ions owing to electrolytic action between the electrically different parts of the iron, oxygen playing an essential part in the process. The ferrous ions then combine with the hydroxyl ions present in the water to form undissociated ferrous hydroxide:

$$Fe + 2OH = Fe(OH)_2.$$

At this stage oxygen plays a second part, converting the ferrous hydroxide into ferric hydroxide or some hydrated form of ferric oxide (rust).

$$4Fe(OH)_2 + O_2 + 2H_2O = 4Fe(OH)_3.$$

The fact that iron rust produced under ordinary atmospheric conditions is shown, on analysis, to contain carbonic acid—that rust formed under such conditions is a basic carbonate of iron—does not necessarily prove, as has been supposed, that carbonic acid plays any essential part in its formation. Wet ferric hydroxide absorbs carbon dioxide, so that the carbonic acid found in rust formed under these circumstances may very probably have been absorbed from the air by the rust after its formation.

It follows from the theory that if we could produce metallic iron chemically and physically homogeneous it ought not to rust, because there would not be any differences of solution-pressure on its surface, and therefore no cause for the production of electric currents when the iron was placed in an electrolyte.

It would follow also from the theory that the production of a coherent and absolutely homogeneous coating, of whatever nature, on the surface of commercial iron would prevent corrosion. The bearing of this on the question of passivity is discussed later.

The Properties of Chemically Pure Iron Considered in Connection with the Electrolytic Theory of Corrosion

The present writer[10] has prepared chemically pure iron, and it has been found that such iron can be exposed to the action of air (or oxygen) and water (either distilled or tap-water) for an apparently indefinite time without showing any signs of corrosion. It must be strongly emphasized, however, that *chemical* purity is not the only essential.

In the preparation of pure iron by the writer's method the same sample of ferric nitrate, treated in exactly the same manner throughout its conversion into iron will not always give like speciments of the metal.

One batch of iron will rust quite readily, while another batch can be exposed for many months to the action of air and water without showing any signs of corrosion. All the pieces of the same batch behave, as a rule, in a precisely similar manner. Now any difference between the batches cannot be due to differences in *chemical* composition. The only possible variable factors are temperature of reduction and rate of cooling, and so differences in the product must be of a *physical* and not a chemical character.

It is a very striking fact that the pieces of iron which will not rust can also be put in solutions of copper sulphate or copper nitrate, of any strength, without causing copper to be deposited on the iron, while pieces from a batch of iron which rusts always cause the deposition of copper from the same solutions of copper salts. Sometimes the copper is deposited quickly, while at other times several hours may elapse before the deposition of copper takes place. The metal is always attacked at one or more points, and the deposition of copper spreads from these points over the whole surface in a very short time.

It is clear from these experiments that physical differences in iron of the same high state of chemical purity can cause most profound differences in its behavior.

If the theory is true, we should expect such results. In the case of the iron which does not rust and is unaffected by solutions of copper sulphate or copper nitrate, the metal is probably physically homogeneous, at any rate on the surface. There would, therefore, be no differences of solution-pressure (no electrically different parts) on the surface of the metal, and so no tendency for the metal to pass into solution by electrolytic action when the metal was put into an electrolyte. Since iron could not, therefore, pass into solution, we should not expect rusting to take place, nor should we expect the deposition of copper on the iron from solutions of copper salts.

The processes preceding corrosion and the deposition of copper from copper salt solutions upon metallic iron are undoubtedly similar in character.

In the case of the samples of pure iron which rust and which cause the deposition of copper from solutions of copper salts, the metal, although chemically pure, is not physically homogeneous, owing probably to some inequality in the temperature of reduction or the rate of cooling.

If such be the case, differences of solution-pressure would exist on the surface of the metal, and when it was put into an electrolyte there would be a tendency for electric currents to be set up between the electrically different points. In water, in the presence of air, this would lead to corrosion, in copper salt solutions to the deposition of copper. An interesting confirmation of such an argument is furnished by the following experiments.

Some pieces of the pure metal were taken which had been exposed to the action of water and air for several months without showing any signs of corrosion. They were carefully dried between pure Swedish filter paper, and then some of them were subjected to pressure in a polished agate mortar

[10] *Trans. Chem. Soc.*, 1910, 97, 2430; 1912, 101, 2068.

by an agate pestle, while others were left untouched. Some of the pressed pieces were then put in distilled water and exposed to the air; they rusted in the course of a few hours, rust forming first at the edges which had not been pressed by actual contact with the agate, while the pressed portions remained quite bright. Others of the pressed pieces were put into a solution of copper sulphate; copper was deposited immediately on the metal, deposition first commencing at the unpressed edges of the pieces.

The pieces of iron which had been dried, but not pressed, neither rusted when placed in contact with water and air again, nor did they cause copper to be deposited from copper sulphate solution, so that it was the application of pressure alone which caused the profound alteration in the properties of the iron.

This again is readily explained by the theory. If a piece of iron be originally homogeneous and part of it be subjected to a strain, then the piece will consist of two parts, a strained and an unstrained part. It will contain two different modifications of iron, each having a different solution-pressure. If such a piece of iron be put in an electrolyte, there would now be a tendency for the passage of an electric current between the two electrically different parts, and so we should expect corrosion to take place when the iron was placed in contact with water and air. The fact that corrosion and the deposition begin at the unpressed part shows that this part is relatively electropositive to the pressed part.

These same points are illustrated in a striking and beautiful manner by the use of the ferroxyl reagent described by Cushman and Gardner.[11] The pure iron which does not rust is unaffected by the ferroxyl reagent, while the pure iron which is not physically homogeneous, or has been subjected to pressure, causes the development of the beautiful colors which indicate electrical differences, or differences of solution-pressure on the surface of the metal.

There are, however, some properties of pure iron which could not be predicted from the theory, but which cannot be considered as arguments against it.

The behavior of pure iron towards solutions of copper chlorine is altogether different from its behavior towards solutions of the sulphate and the nitrate. Pure iron which has remained uncorroded after many months in contact with water and air, or which has remained unaltered in solutions of any strength of the sulphate or nitrate of copper, causes the immediate deposition of copper when placed in a solution of copper chloride, even when the solution is very dilute. The same effect is producted if sodium chloride is added to copper sulphate or nitrate solutions in which the iron has remained unaffected for long periods. The experiment is very striking if carried out in the absence of air. The iron quickly disappears, leaving behind finely divided copper, which in the course of a few hours is itself converted into white insoluble cuprous chloride.

Further, if a tube be taken containing pure iron which has remained unaltered in copper sulphate solution for a long period, and the temperature of the tube be raised to 100 deg. C., copper is deposited on the iron, and in a short time all the iron passes into solution, leaving behind small crystals of copper.

These facts led the writer to suspect that the pure iron might conceivably be covered with a thin coherent film of some protective substance, and most careful experiments were carried out to test this possibility. It was finally decided, as the result of these experiments (*loc. cit.*), that this curious behavior could not be accounted for by the presence of a protective film of either oxide or hydride on the iron.

The curious behavior of the pure iron towards copper chloride solutions is interesting if considered in connection with the behavior of the metal in sodium chloride solution. Even very dilute solutions of the chlorides of the alkali metals have a remarkable action in starting the corrosion of pure iron which has withstood the action of water and air for long periods without showing any signs of corrosion. The action of salt solutions on pure iron is at present under investigation, and the results are not ready for publication.

It is well known that ordinary commerical iron is rendered passive by certain substances, *e.g.*, nitric acid, solutions of caustic alkalies or chromic acid, and that such passive iron, so long as the passivity persists, will not rust in contact with pure water and pure oxygen, and will not cause the deposition of copper from very dilute solutions of copper sulphate.

In view of what has been said above, it seems probable that these substances have the power, in some way or other, of altering the surface of commercial iron so as to destroy, temporarily, the electrical differences on its surface—to produce what might be described as an "electrically equable" surface. The result of this would be that the iron would possess no tendency to go into solution in an electrolyte and so would not rust when placed in contact with air and water.

Much the same sort of "electrically equable" surface is produced in a plate of impure zinc when it is amalgamated.

Since passivity can be produced by such widely different substances, it is not improbable that the "electrically equable" surface produced is not always of the same nature.

The writer is inclined to the view, which is as yet unsupported by experimental evidence, that substances which protect iron from corrosion have the power, in some way or other, of destroying or reducing the electrical differences which always exist in commercial forms of iron—that they have a passiviving effect—and that the substances, like the chlorides of the alkali metals, which stimulate corrosion, do so by increasing these electrical differences. This would explain the remarkable behavior of pure iron towards copper chloride and alkali chloride solutions. For the present, however, the subject must be left open.

[11] *Corrosion and Preservation of Iron and Steel*, p. 48.

A New Sodium and Sodium Peroxide Factory

A new electrochemical company has been formed in Norway under the title of Frederiksstad Elektrochemiske Fabriker, A. S. The capital is $100,000. Mr. I. J. Moltke-Hansen, who was formerly in this country, and later works manager of the Norwegian Ashcroft sodium factory, is to be managing director and general manager. The works will be located at Frederiksstad, Norway. For the present the manufacture of metallic sodium and sodium peroxide will be taken up while the intention is to add soon the manufacture of other compounds. The enterprise is stated to be backed by an important Norwegian financial group.

A New Testing Laboratory

The Abbe Engineering Company, of 220 Broadway, New York City, manufacturers of pulverizing and grinding machinery, vacuum pumps and pressure blowers, have opened an additional testing laboratory for the grinding, mixing, and crushing of samples submitted by prospective customers. The new laboratory occupies the entire ground floor of the building at Huron and West Streets, Brooklyn, N. Y. The laboratory is equipped with pebble mills of various sizes, "Max" mills, cutters, crushers, excelsior mills, vacuum pumps, and pressure blowers.

It is also intended to install an up-to-date bolting reel, on which the different meshes of Swiss silk bolting cloth sold by this company may be used for bolting samples.

The primary object of this laboratory is, of course, to obtain a more thorough and practical knowledge of the particular material to be tested for customers, so that recommendations for machines are based on experimental facts.

The Elmore vacuum process produced 1137 tons of copper concentrate at the plant of the Sulitelma company, Norway, during the month of February, 1913.

Some Effects of Temperature Upon the Resistance of Graphite and Carbon.

By Edwin F. Northrup

In connection with a test upon a graphite electric furnace, which the writer designed for use in his researches upon the resistivities of metals at high temperatures, observations were made upon the resistance of graphite beyond the temperature of melting nickel. The graphite was Acheson graphite of best quality and it was found that this graphite is remarkably constant in resistance up to a temperature of at least 1500 deg. C. The following table is self-explanatory:

W = kilowatts supplied to primary of transformer.
A_s = amperes through furnace.
V_f = volts at terminals of graphite heater.
$\frac{V_f}{A_s}$ = resistance in ohms of graphite heater.

Time P.M.	W	A_s	V_f	$\frac{V_f}{A_s}$	Remarks
4.15	2.5	99.4	20.4	0.205	Room temperature, 23° C.
4.20	2.5	104.0	19.64	0.189	
4.25	2.52	104.8	19.6	0.187	Pb melts, 327° C.
4.30	2.56	105.0	19.6	0.187	Visible red. Zn melted.
4.40	2.6	105.0	20.0	0.190	
Opened switch for 4 minutes.					Sb melted at about 4.35, 630° C.
4.45	2.58	107.0	20.5	0.191	
4.50	2.57	105.8	20.46	0.193	
5.01	2.68	106.4	21.2	0.199	The vapor of boiling Zn obscured the view for a time.
5.08	2.88	109.8	22.1	0.201	Copper starts melting.
5.10	2.92	110.2	22.3	0.202	Copper all melted, 1084° C.
5.20	2.88	108.2	22.5	0.208	
5.30	2.67	103.2	21.8	0.211	
5.36	2.9	107.2	22.8	0.212	Sb boiling.
5.38	2.89	107.0	22.76	0.212	Nickel starts to melt.
5.40	2.89	107.0	22.76	0.212	Nickel all melted, 1452° C.
5.50	2.8	105.0	22.38	0.213	
6.00	3.0	110.0	22.96	0.209	"Lavite" not yet melted ("Lavite" melts at 1525° C.).
6.01	3.0	109.8	22.8	0.208	"Lavite" not melted. Opened switch.

The writer, being aware that ungraphitized carbon (as an electric light carbon) has a marked negative temperature coefficient, devised the following experiment which exhibits the difference between this material and graphite in a striking manner: A rod of graphite and a rod of carbon were selected, each 1¼ in. long and ¼ in. in diameter. These were used to connect together two plates of graphite. Two holes were drilled in each plate of graphite into which were inserted, for a distance of ¼ in., the ends of the carbon and the graphite rods. The graphite plates were connected to the low-tension side of a 2-kw. transformer which yielded, with 110 volts on its primary, 22 volts on open circuit on its low-tension side. A rheostat was put in the primary side of the transformer to limit to an approximately constant value the current delivered by the secondary. On closing the circuit the graphite rod came quickly to incandescence while the carbon rod, in parallel connection with it, remained for a time dark, because of its higher resistance at a moderate temperature. After a fraction of a minute the carbon rod began to heat, and in doing so its resistance rapidly lowered so that it took more current than the graphite rod, and it in turn became brilliantly incandescent, while the graphite rod greatly declined in brilliancy.

The writer concludes that, for small laboratory furnaces at least, graphite makes a better resistor material than ungraphitized carbon. When a constant potential was applied to the terminals of an electric furnace made of Acheson graphite the watt input registered on a wattmeter remained remarkably constant, without any adjustment of rheostats, over a temperature range as great as 1400 deg. C. The regulation of such a furnace is extremely easy.

Palmer Physical Laboratory,
Princeton, N. J.

In sampling ore automatically, time samplers which receive a falling stream of ore for a definite period of time, are regarded as the best devices for the purpose. They operate on the principle of taking the whole stream for a portion of the time, in distinction to that class which attempts to divide the stream of ore and cut part of it all the time. The lack of uniformity in the falling stream renders the latter type of machine inaccurate.

The Uranium and Radium Situation.*

By Dr. Charles L. Parsons.

Some months since rumors reached the U. S. Bureau of Mines of an increased demand for carnotite ores from Colorado and that these ores were being shipped abroad in some quantity. Further, it was reported that the methods of production involved large losses of material and that methods for concentrating low-grade material now being thrown on the dump were greatly needed. Accordingly, Messrs. R. B. Moore and K. L. Kithil were assigned to the task of investigating the situation with headquarters at Denver, where the Bureau established a laboratory for the purpose of investigating the rarer metals occurring in the western part of the United States and problems bearing upon the prevention of waste and increased efficiency in the mining industry. The surprising conclusion has been reached that while all the radium placed upon the market in the last few years has been produced in Europe, a large portion of this output has come from American ores.

Radium institutes have been established in Austria, France, Germany and England, a European science and industry have been developed from American radium ores and even the uranium present with the radium has been manufactured into marketable condition only in foreign countries and returned in finished condition to our own. American hospitals and physicians have been forced to procure from abroad such radium as they could afford for experimental purposes, and investigations in our governmental and university laboratories of the wonderful properties of radium and their possible application to the eradication of disease and the development of industry have been hampered by the almost prohibitive prices at which the finished material is held.

While the Austrian government, realizing the untold possibilities of the radium ores of St. Joachimstaal, has purchased the mines, put their output under direct governmental supervision, and has entered into an arrangement whereby this ore is worked up in cooperation with the Vienna Academy of Sciences for experimental purposes in a carefully administered radium institute, America has allowed her large and much greater resources to be exploited on a basis which wastes perhaps irretrievably a large portion of the material mined, and has exported carefully selected ores at a price by no means commensurate with its radium value if worked up at home.

Even before carnotite was exported, pitchblende of the highest grade was sent out of the country at the time when the world's radium output was supposed to be coming from Austrian ores. At least 20 to 25 tons of high-grade pitchblende has been sent out of the country. Within the last two years, however, foreigners have realized the value of our carnotite resources and most of the radium that has been exported has gone abroad in this ore.

During the last year, carnotite carrying 28.8 tons of U_3O_8, from which 8.8 grams of radium chloride or 11.43 grams of radium bromide could be obtained, were produced. Practically all of this ore was shipped abroad for the extraction of radium. The value of the radium salts extracted would be at the minimum market price $528,000. The total supply of radium salts from all other sources including the Austrian mines was probably not more than 3.65 grams of radium chloride, basing the production of the Austrian mines for 1912 upon that of 1911 which is known.

Pitchblende, the richest of all uranium minerals, is composed mainly of uranium oxide, but also carries lesser quantities of a large number of other substances. It has been found in small quantities in Connecticut and in the feldspar quaries of North Carolina. Practically the total American output has come from the mines in Quartz Hill, Gilpin County, Colorado. The mineral is a heavy black substance which can be readily identified by any one who will suspend a sample of the pitchblende above a photographic plate wrapped in black paper and kept in the dark for a few days with a key or other metal opaque to radium radiations placed between the sample of ore and the

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plate so that when the plate is developed a shadowgraph of the object may identify the ore. Pitchblende may carry as high as 80 per cent. uranium oxide, although the average ore is not nearly as rich.

Carnotite is a yellow mineral consisting mainly of potassium uranyl vanadate but containing also small amounts of barium and calcium compounds. Being a uranium mineral as is pitchblende it of necessity carries radium, although it has not yet been definitely established that the uranium and radium are in equilibrium as they are in pitchblende. However, it is known that in our western carnotite the amount of radium is not far from the equilibrium ratio and in calculations given above an allowance of 10 per cent has been made to cover this possible deficiency. While carnotite is known to occur in smaller quantities in other states, the more important deposits are scattered over a considerable area in Colorado and Utah, embracing Meeker and Skull Creek, Colo., Green River, Thompson's Moab, Richardson, Table Mountain, Pahreah, and other places in Utah. The largest proportion of the ore, however, has been produced in or around Paradox Valley in southwest Colorado from which it has to stand long hauls by pack animal or wagon to the railroad. Carnotite always carries vanadium as well as uranium and radium, but is purchased almost wholly on its radium content, comparatively little being allowed for the vanadium present.

The ore consisting of a fine grained sandstone containing yellow finely pulverulent carnotite occurs in pockets and is easily mined. As ore below 2 per cent uranium oxide cannot at the present time find a market, a considerable portion of the ore has been thrown on the dump and is now being wasted as material of lower grade has to be discarded on account of the long haul and the fact that European buyers have set this standard as to quality. Ores of higher grade are sometimes obtained, but they occur only in small pockets and it is generally advisable to mix these high-grade ores with ores of somewhat lower grade in order to increase the marketable output. Ore of 2 per cent uranium oxide is now worth approximately $75 per ton f.o.b. New York.

In the mining of these carnotite ores it is probable that 5 tons of material capable of concentration are thrown upon the dump for every ton that finds its way to market. To develop methods for concentration of these ores and save the valuable material now wasted is one of the problems before the Bureau of Mines with fair prospect of a successful conclusion.

It is difficult to estimate the total amount of radium that has been produced up to the present time, but it is quite certain that if the ores which have been mined in this country and abroad and sold for radium production have been actually worked up into this material there is now in existence something like 40 grams (1¼ ounces) of radium. The price of radium salts varies somewhat. In large quantities it has been $60,000 per gram for both radium chloride and radium bromide, although the latter contains less metallic radium in proportion to its weight than the former. It should be remembered, therefore, that it is more advantageous to purchase radium chloride than radium bromide. In small quantities the average price has been $80,000 per gram which represents about $2,500,000 an ounce.

The figures given show very plainly that the United States has taken the palm from Austria as the radium-producing country of the world. Very few people have been cognizant of the fact that the United States has such deposits within her borders. Up to the present time very little interest has been taken in the matter and only one firm has engaged in the extraction and refining of radium in this country—a condition which is deplorable. This firm has not yet entered the radium market.

Practically every ton of ore mined in 1912 went abroad, and as the American deposits are far from being inexhaustible we are rapidly depleting our own reserve and are shipping from the country material of great value and of unknown possibilities which cannot be replaced.

The applications of radium are still too little understood to admit of definite statement. Its discovery and marvelous properties have already changed our ideas regarding the constitution of matter and scientific investigation will undoubtedly lead to valuable results which we cannot now even foresee. Altogether too many incorrect statements and vague speculations have been placed before the public as to its use in medicine. A recent report of the London Radium Institute and the many articles emanating from minor laboratories experimenting in the application of radium to therapeutics all tend to show, however, that it has a real value, the certain application of which must await further experimentation. In the meantime no credence should be given to the many stories that are sure to be printed unless they are backed up by the highest medical authority, which will always give publicity with caution.

The best medical authorities appear to agree that up to the present time radium has not been proved to be specific for any disease, although it has been shown to be helpful in many cases and the outlook for its future application to certain diseases not easily treated otherwise are decidedly encouraging.

Apparently no uranium is worked up in the United States, but according to statistics gathered by the division of mineral resources of the United States Geological Survey about $14,000 worth of its oxides and salts were imported into the United States in 1911. It is one of the few materials shipped abroad as ore and returned in manufactured form.

A preliminary report on Uranium, Radium and Vanadium, by R. B. Moore and K. L. Kithil, will soon be issued by the Bureau of Mines. This bulletin describes the carnotite deposits of Colorado and Utah and the pitchblende deposits of the former state. It also contains detail of which the foregoing is simply a general summary which cannot fail to be of value to all those interested in our mineral resources and their development.

Division of Mineral Technology,
U. S. Bureau of Mines.

The mineral production of Canada in 1912, according to preliminary estimates by the Canadian Department of Mines, shows an increase of nearly 29 per cent as compared with that of 1911. Porcupine was the only new contributing district of any importance, with a gold production of $1,750,000. There was an increased output in all the provinces during the year. Copper and asbestos contributed to the increase in Quebec; nickel, copper and gold in Ontario; coal in Alberta; gold, silver, copper, lead, zinc and coal in British Columbia. The production of cement is interesting; the Canadian output increased over 26 per cent; the imports, 116 per cent, and total consumption, 34 per cent.

Paraffin Paper for Spot-Plates.—The Mine & Smelter Supply Company, Denver, has recently devised a substitute for the ordinary porcelain spot-plate for laboratory use. It consists of a pad of paraffined sheets of white paper, from which a sheet may be torn when it can no longer be used. The paraffined surface causes the spots to assume a spherical form, instead of spreading and running together as is likely to be the case with porcelain surfaces.

Potash Lands Withdrawn.—Three tracts of land in the desert basins of California and Nevada have been withdrawn from entry, as they are believed to contain valuable deposits of potassium salts and brines. The national government is at the same time making provisions to relieve the present chaotic condition regarding such lands, and will endeavor to provide a safe legal basis for the development of the deposits now known and any that may be developed in the future.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in January, 1913, was sixty-one. The quantity of ore milled during that period was 2,347,929 tons. There were 10,009 stamps in operation with an average duty of 8.69 tons per twenty-four hours. Tube mills in commission numbered 286. The yield for the month was 789,390 fine ounces gold. The yield per ton of ore milled was 6.56 dwt.

Atlantic City Meeting of the American Electrochemical Society

The twenty-third general meeting of the American Electrochemical Society was held in Atlantic City from April 3 to 5 and was a most enjoyable affair. There was a good attendance, the registry list containing over 150 names (all coming from other cities). There was ideal crisp spring weather, just right to enjoy Atlantic City at its best. There was a series of very interesting papers on various electrochemical subjects, and a most instructive symposium on electrodeposition of metals. There were lively discussions in open meeting and quietly between man and man. And above all there reigned supreme the old and ever young happy congeniality which is typical of every Electrochemical Society meeting.

Thursday, April 3, was devoted to the annual business meeting and the reading and discussion of papers on various electrochemical subjects. On Friday an excursion was made to Philadelphia, while a most interesting experimental lecture was delivered in the evening by Professor Kenrick. The two sessions of Saturday were devoted to the presidential address of Prof. W. Lash Miller and to the symposium of papers on electrodeposition.

PHOTOGRAPH TAKEN AT THE SATURDAY SESSION OF THE ATLANTIC CITY MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY

Excursion to Philadelphia

Early on Friday morning the members of the Society were taken by a special car to Philadelphia and from there to Lansdowne, Pa., where the works of the Crucible Steel Casting Company were visited. Besides the oil-fired crucible furnaces the feature of principal attraction was the two-ton Roechling-Rodenhauser induction steel furnace (the only one of this type in the United States). This was shown in operation and explained by Mr. Cunningham, Mr. vom Baur and Herr Schoenawa. It is, however, not run now as a Roechling-Rodenhauser combination furnace, but as a pure induction furnace. No current is supplied to the bath through the pole plates, which have been disconnected in order to improve the cooling action by removal of the winding which fed the pole plates when originally installed.

The party then went to the University of Pennsylvania, where the Society was entertained at luncheon. The provost of the University, Dr. Edgar F. Smith, heartily welcomed the Society to the University in a felicitous speech. For the Society the president-elect, Dr. E. F. Roeber, replied, emphasizing the debt of gratitude which electroanalysis and electrochemistry in America owes to Dr. Smith and reminding the audience of the fact that it had been in Dr. Smith's chemical laboratory that the original meeting of the Society had been held eleven years ago.

In the afternoon two different parties were formed. One visited the works of Harrison Bros. Company, on Gray's Ferry Road, where lead paints, lithophone, and contact sulphuric acid are manufactured, while the other party was taken by automobiles to the United Gas Improvement Company's works at Point Breeze. Afterward all met again at Broad Street Station and returned to Atlantic City.

The hotel headquarters were at the Hotel Traymore, where all professional sessions were held in the solarium. No official dinner or smoker had been arranged for the meeting. But section Q held several informal sessions in the "Icebox" and elsewhere.

Business Meeting

The annual business meeting of the Society was called to order in the morning of Thursday, April 8, by the President, Dr. W. Lash Miller, of the University of Toronto. From the report of the Secretary it was learned that the Society is in

a flourishing condition, the total of the investment funds being more than $6,000 and the membership on December 31, 1912, being 1350.

The result of the latter ballot for election of officers for the next year was then announced.

Dr. E. F. Roeber is the new president.

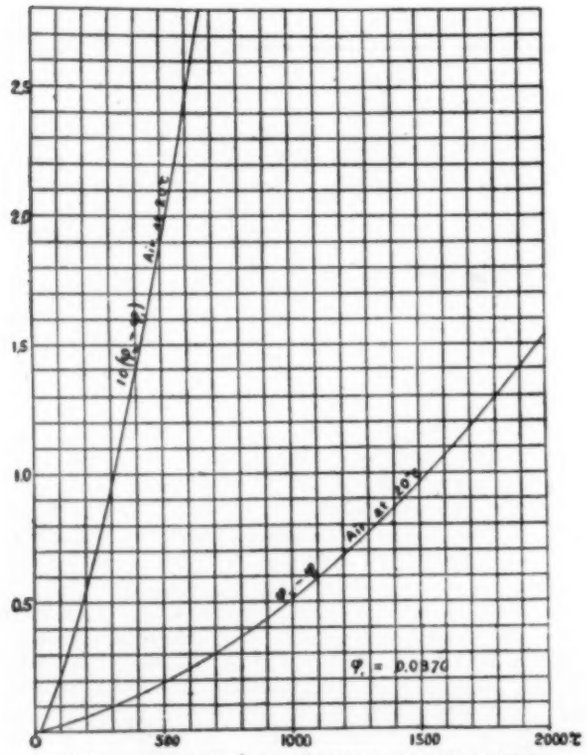


FIG. 1.—CURVE OF ϕ AS FUNCTION OF TEMPERATURE

Dr. Joseph W. Richards and Mr. Pedro G. Salom were re-elected secretary and treasurer respectively.

Prof. C. F. Burgess, Prof. S. A. Tucker, and Mr. C. P. Townsend were elected vice-presidents, and Mr. F. A. J. Fitzgerald, Dr. J. W. Brown, and Dr. C. G. Schluederberg managers.

Convection and Radiation of Heat

Dr. **Irving Langmuir,** of the Research Laboratory of the General Electric Company, of Schenectady, N. Y., presented two papers on the convection and radiation of heat. The chief results of the first paper are as follows:

The total energy loss from a heated surface in a gas consists of two parts, convection and radiation.

If the emissivity of the surface is known the radiation can be calculated from the Stefan-Boltzman law.

$W_r = 5.9 \times 10^{-12} e\,(T_2^4 - T_1^4)$ watts per sq. cm. (3)

e is the emissivity of the surface, and varies for different surfaces from 0 to 1. For any one surface it is accurate enough for most purposes to consider it independent of the temperature.

Free convection from a surface consists essentially in conduction through a film of gas of definite thickness, and can be calculated by the ordinary laws of heat conduction when the film thickness is known.

The loss by convection is equal to the product of two factors, thus:

$$W_c = S\,(\phi_2 - \phi_1)$$

S is the shape factor, and depends only on the shape of the body and on a quantity B, a constant for any given gas under given conditions of temperature and pressure. B is equal to the thickness of the film of relatively stationary gas through which conduction takes place in the case of convection from a plane surface $\phi_2 - \phi_1$ is equal to $\int_{T_1}^{T_2} k\,dT$ where k is the heat conductivity of the gas. The heat conductivity can be readily calculated from the specific heat and viscosity of the gas. Tables of ϕ for air, nitrogen, hydrogen, carbon dioxide and mercury vapor are given. A curve of ϕ as a function of the temperature is given for air in Fig. 1.

The shape factor S has been calculated by the author of the case of convection from planes, cylinders and spheres.

In this way the following formulas for convection are derived:

For Plane Surfaces:

$$W_c = \frac{\phi_2 - \phi}{B} \text{ watts per sq. cm.}$$

For Horizontal Cylinders of Diameter a:

$$W_c = S\,(\phi_2 - \phi_1) \text{ watts per cm.}$$

where S is found from a certain equation given in the paper. But it may be more easily determined as follows:

In Table I S as a function of a/B is given. If a curve is prepared from this data the value of S may be found in any particular case with great ease.

TABLE I.—SHAPE FACTOR FOR WIRES

S	a/B	S	a/B	S	a/B	S	a/B
0.0	0.0	5.0	0.453	10	1.696	30	7.738
0.5	0.56 x10^{-3}	5.5	0.558	12	2.263	32	8.370
1.0	0.594 x10^{-2}	6.0	0.671	14	2.844	34	8.995
1.5	0.725 x10^{-2}	6.5	0.788	16	3.438	36	9 622
2.0	2.752 x10^{-2}	7.0	0.908	18	4.040	38	10.25
2.5	0.0644	7.5	1.032	20	4.645	40	10.87
3.0	0.1176	8.0	1.160	22	5.263	42	11.50
3.5	0.185	8.5	1.291	24	5.877	44	12.14
4.0	0.265	9.0	1.424	26	6.505	46	12.77
4.5	0.354	9.5	1.561	28	7.122	48	13.40
5.0	0.453	10.0	1.696	30	7.738	50	14.03

For Spheres of Diameter a:

$$W = S\,(\phi_2 - \phi_1) \text{ watts}$$

where S is found from the equation:

$$\frac{1}{a} - \sqrt{\frac{\pi}{SB}} = \frac{2\pi}{S}$$

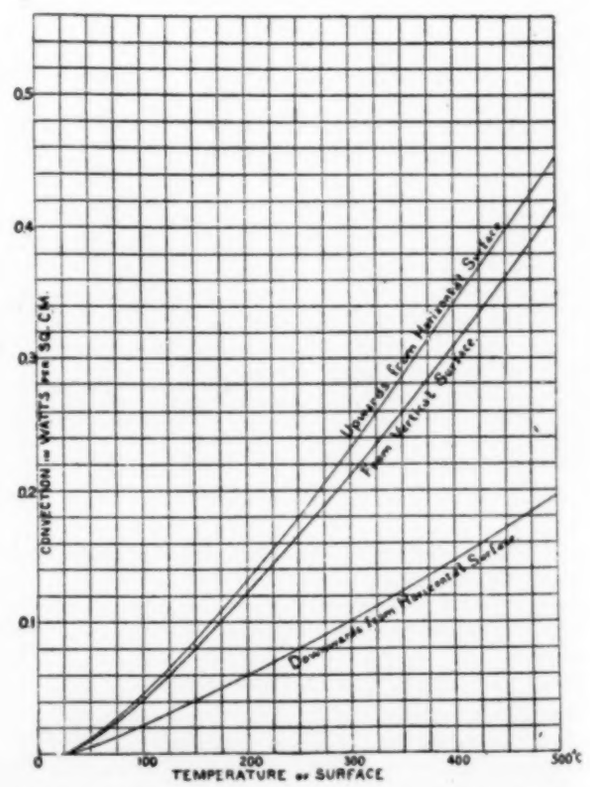


FIG. 2.—RADIATION FROM HORIZONTAL AND VERTICAL SURFACES

In each of these cases the convection can be calculated if we know $\phi_2 - \phi_1$ and B the film thickness for a plane surface.

For air at atmospheric pressure and room temperature, experiments on wires of various sizes and on plan surfaces

give B independent of the temperature of the heated body. The experiments on wires gave $B = 0.43$ cm. and the experiments on vertical plane disks gave

$$B = 0.45 \text{ cm.}$$

From horizontal plane surfaces the convection from the upper surface is about 10 per cent. greater than from a vertical surface, but from the lower surface about 50 per cent. less than from the vertical surface (Fig. 2).

The emissivity of various metallic surfaces was found to be:

Black Body	100
Copper, oxidized	72
Copper, calorized	26
Silver	3
Cast iron, bright	22
Cast iron, oxidized	62
Aluminum paint	50
Gold enamel	37
Monel metal, bright	43
Monel metal, oxidized	43

The total surface losses from these metals at various temperatures up to 600° C. are given in a table in the paper. In

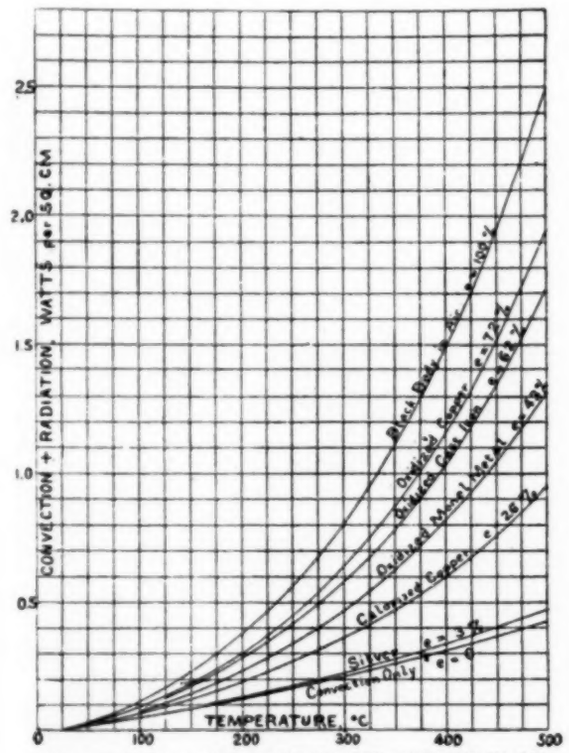


FIG. 3.—TOTAL LOSSES FROM HEATED SURFACES IN AIR

the curves of Fig. 3 are given plots of the total energy losses calculated by adding the radiation (from Stefan-Boltzman law) and convection (from $(\phi_2 - \phi_1)/B$).

Dulong and Petit's law and Lorenz's law for the relation between convection and temperature of the hot body hold very satisfactorily up to very high temperatures. At low temperatures these equations probably give the convection even more accurately than those calculated from the film theory, but at high temperatures the film theory gives much better results. Furthermore, the newer theory enables the effect of the shape and size of the body to be taken into account.

For forced convection, that is, loss of heat in strong air currents, the film theory does not seem to apply. On the other hand, an equation derived by A. Russell seems to agree well with the experimental data available.

For other gases than air, or for air at other temperatures or pressures, the value of B can be assumed to be inversely proportional to the density of the gas. This will give results usually accurate enough for the calculation of convection from small wires.

The second paper by Dr. Langmuir deals with the calculation of the shape factor of solid bodies and gave especially a practical formula for rectangular furnaces.

In the discussion Dr. W. L. Miller referred to the analogy between heat diffusion and diffusion in liquids.

Dr. Hering, in discussing Dr. Langmuir's film on the surface of a body giving off heat, referred to the reverse phenomenon, namely, the importance of the film on the surface of a body which receives heat, like a boiler tube. A single postage stamp pasted on the outside of a boiler tube is not charred. But if enough stamps are pasted one above the other, the last one will be charred. As to the enormous difference between the emissivities of different substances, it might look ridiculous at first glance to silverplate the outside of an electric furnace in order to reduce its radiation loss, but it was certainly worth while to take the emissivity of the furnace walls into consideration, as it would permit the use of thinner walls.

Dr. Richards also emphasized the importance of keeping the outside surface of a furnace clean and polished. If the radiation from the outside surface is reduced, its temperature rises and the loss of heat by conduction through the walls is also reduced on account of the reduced temperature difference inside and outside.

Experiments with Furnace Electrodes

A paper on this subject was presented by Mr. **F. A. J. Fitz Gerald,** of the Fitz Gerald & Bennie Laboratories, Niagara Falls, N. Y., and Mr. **A. T. Hinckley,** of the National Carbon Company, Cleveland, Ohio. Some years ago the National Carbon Company commenced an extensive series of experiments for the purpose of finding the best methods of making large carbons for the larger sizes of electric furnaces coming into use so as to obtain a thoroughly trustworthy electrode that would not fail under the severest usage and at the same time give the smallest wear under the particular conditions which surrounded its use. This study also involved the problem of joining electrodes together so as to avoid the loss in "butts."

The authors describe in their paper the methods used in testing the experimental electrodes and some of the results recently obtained.

The **real density,** that is, the density of the material free from pores, may be considered as a measure of the chemical composition of the material, and increases as the amount of hydrogen in the hydrocarbons decreases. It is thus an indication of the relative rate of oxidation of the electrode, it being a general rule that hydrocarbons high in hydrogen are most readily oxidized.

The determination is made by grinding the sample to such a fineness that all pores are open to the liquid medium. For ordinary carbon electrodes particles passing through a screen having 100 meshes to the linear inch are sufficiently fine. Carbon ground to pass through a 200-mesh screen does not show any increase in the second decimal place in the density determination. The weighed sample is placed in a small specific-gravity bottle with kerosene oil of known density, and evacuated to 5 mm. of mercury so that the pores are filled with the oil. The bottle is then filed with the oil, brought to a given temperature in a thermostat and weighed. The precision of the measurement is 0.5 per cent.

The **apparent density,** that is, the density of the carbon including the pores, may be considered as a characteristic of the mechanical structure. It is influenced by every process through which the carbon goes in the manufacture of the electrode, and is in general a measure of the relative rate of disintegration of the electrode in use. For large electrodes the most satisfactory method of determination is to weigh the whole electrode and calculate its volume from its dimensions. To secure a precision of 2 per cent the weight and length should be measured to 1 per cent and the diameter or periphery to 0.5 per cent.

The **electrical resistivity** of carbon electrodes varies with both the real and apparent densities. Its determination is not

necessary so much to show the physical properties of the carbon as to provide a means of comparison in electrical terms. With large electrodes its determination is made most readily by passing direct current through the whole carbon and measuring the potential drop through a known distance. For ordinary purposes it is sufficient to measure the resistivity to 0.0001 ohm, a precision of about 5 per cent. This means, approximately, that the volts, amperes and length must be measured with a precision of 1 per cent and the diameter or periphery with a precision of 0.5 per cent. If these rules are followed the magnitude of the testing current is of no importance, provided that it is not sufficiently great to heat the carbon appreciably. Care must be taken in selecting the points for the potential contacts that the distance between them and the current contacts is so great that the equipotential surfaces are substantially plane surfaces at right angles to the axis of the electrode. The resistance of the potential contacts may be practically eliminated by drilling small mercury cups in the carbon or pressing the contact pins against the carbon at a pressure of about 3000 lb. per square inch (210 kg per square centimeter) when using a millivoltmeter having a resistance of 1.07 ohms.

Service Tests.—Useful as the laboratory tests are on electrodes, they do not give sufficient information as to what may happen when the electrode is put in actual service. A serious objection to some of the earlier carbon electrodes used

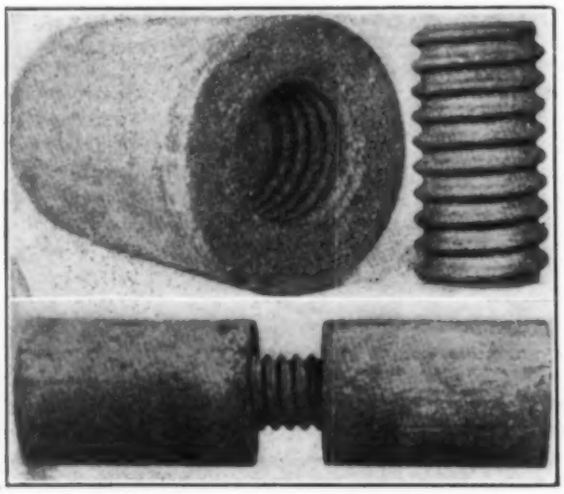


FIG. 4.—JOINING CARBON ELECTRODES

in electric furnaces was the large percentage which broke in service.

Another important consideration in furnace electrodes was the life, some electrodes giving apparently much longer service than others. It is not implied that the astonishing variation in the figures for electrode consumption that is found in technical literature is due altogether to differences in the electrodes used, for this is in large part explained by the irrational method of estimating electrode consumption. For example, however useful from a purely commercial point of view the method of estimating electrode consumption in pounds per ton of steel may be, the important consideration when comparing electrodes is to determine how many kilowatt hours are generated in the furnace per unit weight of electrode. Studied from this point of view it is found that great variations exist.

In experimenting with the manufacture of carbon electrodes it is, therefore, necessary actually to use them in an electric furnace and thus test their behavior under working conditions.

Finally, in working on the problem of making electrodes which can be joined together, laboratory tests of the electrical resistance of the joint, or even tests of its mechanical strength, are not sufficient. It is necessary to use the jointed electrode in a furnace and determine just how it behaves throughout a series of runs.

The authors describe the tests to which the electrode was submitted in an open-arc furnace of the Siemens type, where it worked above a molten bath.

In making a test the furnace was charged with cast iron and steel borings, and the electrode lowered until an arc was started. Then, if it was simply desired to find out if the electrode would stand severe usage without breaking, the power, which should normally have been 150 kw, was increased to 200 kw or 250 kw. In general it is advisable to use an electrode, in a furnace of this sort, with a current density of not more than 40 amp per square inch (6.2 amp per square centimeter), but occasionally higher current densities are necessary or desirable, so that the electrode must be constructed to stand these. Thus, in the experiments to determine the resistance under severe conditions, current densities of 50 amp to 65 amp per square inch (7.8 amp to 10.1 amp per square centimeter) were used. Moreover, for short periods the current density was as high as 77 amp per square inch (11.9 amp per square centimeter).

In making tests of carbon consumption it was soon found that in order to get consistent results the furnace must always be run under identically similar conditions. Before beginning a series of heats the electrode was carefully weighed. Then the furnace was charged with the cast-iron and steel borings, these melted down and the furnace tapped. After a certain number of heats, usually eight, were run off the electrode was again weighed, and the total kilowatt hours used was divided by the loss in weight of the electrode, which gave the kilowatt hours per pound of electrode consumption.

As a result of numerous experiments it has been found that the joint best suited to the average furnace needs is made by using a cylindrical threaded plug, which is screwed equal distances into the ends of the two electrodes to be joined. A shallow rounded thread proves better than a pointed one, or even a truncated one, as there is less tendency to breakage during assembling. A specially prepared paste is used to insure good electrical contact, but this does not interfere in any way with unscrewing the joint when it is desired to replace a section. Fig. 4 shows this method of joining electrodes.

In order to determine the resistance of the jointed electrode to severe usage the method followed was just the same as in the case of the ordinary electrode as regards overloading; but in addition to this a long series of heats were made so that the behavior of the joint as it finally worked down to the bath could be noted.

A very important consideration in the case of jointed electrodes is the resistance of the joint. The method of measuring the resistance of the joint is described in detail.

A large number of electrodes which had gone through different manufacturing processes were tested in the furnace simply for the purpose of determining whether or not they would break in use when subjected to the severest conditions, such as overloading, sudden changes of temperature, etc., and as a result much knowledge has been gained as to the causes of electrode breakage.

In order to study the influence of different manufacturing processes on the consumption of carbon under definite conditions of use a large number of electrodes were manufactured, these being divided into several series. Only one of the numerous factors entering into the manufacturing process was varied in any series, every other condition being kept the same. When the electrodes were made they were tested by the regular laboratory method, and then submitted to the service tests described above. In this particular investigation there have been six series of carbons tested.

As regards the real density no variations have been found, all electrodes giving the same results. As the attempt was made so to adjust conditions of manufacture that there should be no variation in this physical characteristic the results may be considered as highly satisfactory, and give reasons for believing that the great variations found in other characteristics were due simply to the deliberate differences in manufacturing process and not to some unknown causes.

Considerable differences were found in the apparent densities of the electrodes tested, these varying from 1.40 to 1.50. The most interesting results, however, were those obtained in determining the kilowatt hours used per unit weight of carbon, for it was found that the extreme variations were as much as 100 per cent. The importance of this discovery can scarcely be overestimated as showing the great influence manufacturing conditions may have on electrode economy. Such astonishingly large variations were at first viewed with suspicion, for it was feared that they were really due to the way in which the electrodes were used in the furnace and not to inherent peculiarities of the carbon. Experiment showed, however, that when tests were repeated the results checked in a satisfactory way.

In testing the jointed electrodes a great number of heats were made so that the behavior of the joint itself when in the furnace could be noted. Fig. 5 is from a photograph showing a partially consumed joint. In one set of tests an electrode was made by joining four 15-in. (380 mm) sections together, thus making an electrode 60 in. (1520 mm) long and 10 in. (254 mm) in diameter. The electrical connection for the cables was made at the top section, and new sections added to the electrode as it wore away. With this electrode sixty heats were made, the behavior of the electrode throughout being excellent.

From several tests, details of which are given in the paper,

FIG. 5.—CONSUMPTION OF ELECTRODE

it is seen that the resistance of the joint decreases as the electrode gets hotter. It appears that the heating of the joint improves the contact between the sections and the connecting plug.

In the discussion which followed Mr. Petinot referred to the difficulties of joining electrodes in actual furnace practice. At a joint (as they were formerly made) it sometimes happened that all the current passed through the nipple which thus became very hot, while the outside of the electrode at the joint was cold. It may be advisable to make the nipple of graphite, but if satisfactory electrodes 10 ft., 15 ft. or 20 ft. long could be made, they would be better than any method of joining.

Dr. Richards advised to use as long electrodes as possible and make the connection as close to the furnace as possible.

Mr. Hansen agreed that long electrodes are very nice with stationary furnaces, but they may become a nuisance with tilting furnaces. As to the electrode consumption, he pointed out the importance of oxidation by the slag.

In reply to a question by Dr. Whitney as to electrodes with mineral materials, Mr. Hinckley replied that certain mineral materials increase the life of electrodes. The paste used in the joint is a graphite mixture.

Mr. Lidbury said that the most important point in connection with electrodes is that they should not break under "reasonable treatment." But as Mr. Fitzgerald said, there is a difference of opinion between manufacturers and users of electrodes as to what is reasonable treatment.

Aluminium Nitride.

A paper on aluminum nitride and the Serpek process, by Dr. **Jos. W. Richards,** of Lehigh University, was the first paper of Thursday afternoon. It was essentially the same paper as presented by the author before the New York section of the American Electrochemical Society and published in full in our March issue, page 137.

New, however, were the following notes on the Serpek reaction.

The decomposition of alumina by carbon in presence of nitrogen to form aluminium nitride is a reaction which is interesting to study and compare with analogous reactions.

Professor Baron, in 1760, tried to reduce alumina by carbon, but was unable to obtain the temperature required.

Oerstedt, in 1817, devised a way to decompose alumina by carbon, finding that this took place in a stream of chlorine gas, at a low temperature, the products being aluminium chloride and carbon monoxide gas.

$$Al_2O_3 + 3C + 3Cl_2 = 2AlCl_3 + 3CO.$$

This reaction is practically the exact analogue of the Serpek reaction, using chlorine instead of nitrogen. Deville, for instance, who ran this operation on a large scale, remarks that the retort is heated to a uniform dark-red heat (which would be about 600 deg. C.) and that "however fast the chlorine may pass into the retort, it is so well absorbed during three-fourths of the operation that not a trace is mixed with the carbon monoxide escaping."

If the query be made as to why the reaction is so easy, thermochemistry gives us the answer: the reaction is exothermic 17,880 calories.

$$Al_2O_3 + 3C + 3Cl_2 = 2AlCl_3 + 3CO.$$
$$-391{,}600 \qquad +322{,}000 + 87{,}480 = +17{,}880$$

It is profitable to consider in this connection another analogous reaction.

$$Al_2O_3 + 3C + 3S = Al_2S_3 + 3CO.$$

This reaction does not take place at 1200 deg. C., and we find it would be endothermic (heat absorbing) 180,200 calories. However, the reaction with carbon and sulphur already combined as CS_2 does take place, readily, at 1000 deg. C., and this reaction is much less endothermic than with C and S separately:

$$Al_2O_3 + 3CS_2 = Al_2S_3 + 3COS$$
$$-391{,}600 + 87{,}000 + 124{,}400 + 111{,}090 = -69{,}110$$

Coming now to the nitrogen reaction, we have:

$$Al_2O_3 + 3C + N_2 = 2AlN + 3CO$$
$$-391{,}600 \qquad +90{,}900^* + 87{,}480 = -213{,}220.$$

The figure 90,900 was given as provisional. It is not a measured value.

The above is therefore a highly endothermic reaction and corresponding with that we have the fact that it does not take place to an appreciable extent below 1500 deg. C. Aside from all thermal losses, we can say, therefore, that the reaction itself absorbs 213,220 calories = 248 kw-hours for 82 kg. of aluminium nitride formed; which equals 3000 kw-hours = about ⅓ kw-year per ton of nitride. At least double this amount would be required in practice.

Tests of the Edison Storage Battery

"Some Tests of the Edison Storage Battery" was the title of a paper presented by Messrs. **C. W. Bennett** and **H. N. Gilbert,** of Cornell University. The object was to run a few efficiency tests at different rates of discharge, study the effects of charging at different rates, check the measurements of capacity, and study the cell operating at a low temperature.

The following conclusions are drawn by the authors:

1. The energy efficiency, at the normal rate of operation, is about 55 per cent, the amp-hour efficiency being about 85 per cent.

2. The cell is capable of operating continuously at high rates.

3. At the 40-minute discharge rate, with charge at the normal rate, the energy efficiency is about 32 per cent, the amp-hour output being about 60 per cent of the rated output.

4. The cell can be charged at high rates, the efficiency being about normal when a "booster" charge at 3⅓ times the normal rate, for one hour, was given the cell. There is no reason why the cell should not be charged at five times the normal rate for a few minutes.

5. If the cell is to be charged the day before it is to be used, energy is conserved if the charge is maintained until the voltage reaches about 7 volts, instead of carrying it to completion. In other words, no nickel peroxide should be formed on charging, the loss of energy on standing being due presumably to the spontaneous decomposition of this compound.

6. If the charge is stopped before the higher oxidation products of nickel are formed, the efficiency is higher than normal being about 64 per cent.

7. At 4 deg. C. the battery is capable of delivering more than two-thirds the normal output, when charged at the same temperature. The voltage, of course, is lower than normal, for the internal resistance of the cell is higher at the lower temperatures. The efficiency also is low.

8. The cell may unquestionably be allowed to stand an indefinite length of time without injury.

9. The flexibility is so great that the cell can be used under the most adverse circumstances.

10. The method of charging until the voltage has remained constant for one hour gives satisfactory results, but requires watching or the use of a recording voltmeter.

11. The continuous drop in voltage on discharge (about 10 per cent per hour) is a disadvantage against this cell, over the lead cell.

12. When referred to equal amounts of power delivered, the Edison battery weighs about 25 per cent less and costs about 25 per cent more than the Iron-clad Exide battery.

The paper was discussed at some length by Messrs. Clymer, Langmuir, Hering, Smith, Calmus, Schluederberg, and Smith. Mr. H. H. Smith, of the Edison Storage Battery Company, said that the Edison battery had been placed on the market in a radically changed structure in 1908, and for this reason the literature prior to 1908 could not be relied upon as referring to the present Edison battery. He also said that the paper was based on isolated tests and that there were some inaccuracies and some discrepancies between the drawings and the tables. He criticised, especially, the voltage figures and said that the established average discharge voltage of the Edison battery at normal rate was 1.20 volts per cell. He thought that conclusion 5 of the authors was not substantiated by evidence in the paper and was merely an assumption and that conclusion 11 was hardly accurate. He also claimed that the use of recording meters was not reliable.

Dr. Bennett in his reply acknowledged some inaccuracies in the drawings which were due to a misunderstanding of the draftsman, but said that the figures in the tables were correct, though no very high accuracy was claimed for them, as had been clearly stated in the paper.

A New Type of Concentration Cells

A paper by Dr. **Reinhard Beutner,** of the Rockefeller Institute for Medical Research, New York City, discussed "concentration cells containing organic liquids immiscible with water."

A year ago he presented a paper before this Society on electromotive forces generated by systems containing immiscible electrolytic conductors. (This journal, Vol. X, p. 297.) He mentioned in that paper some investigations relating to biological applications of the electrochemical researches along this line carried out by him and Dr. J. Loeb, in which it had been found that the skin of plants behaves like an electrode reversible with respect to various positive ions.

As he pointed out, a phenomenon of this kind was not yet known in electrochemistry, though certain investigations of Nernst and of Haber (which were there discussed at length) had shown that a phase junction between immiscible electrolytes acts like the surface of a metallic electrode. This did not explain how such an effect was possible.

More recent investigations which the author has carried out have proved that with a number of water-immiscible organic substances similar effects can be produced as with the skin of plants. This has led to further investigations into the electromotive properties of water-immiscible organic substances, and the surprising result has been reached that cells may be built up from such organic compounds and aqueous solutions showing an electromotive force which nearly equals in magnitude the e.m.f. of galvanic cells with metallic electrodes. An explanation of these phenomena has been found possible on the basis of the well-known theories of chemical thermodynamics. The quantitative mathematical applications involved, however, many difficult questions.

The results of the experiments and theoretical researches of the author are given in four chapters, dealing with salicylic aldehyde as electrode, toluidine as electrode, experiments with mixtures of toluidine and salicyclic acid which explain the nature of the e.m.f. under discussion, and e.m.f's. due to chemical reactions as distinguished from e.m.f's. due to mere concentration differences.

The president, Dr. Miller, congratulated the author on the ingenuity of his indirect methods of checking his results.

Concentration Changes in the Electrolysis of Copper Sulphate Solutions

A paper on the above subject, by Messrs. **C. W. Bennett** and **C. O. Brown,** of Cornell University, was presented by Dr. Bennett.

In the electrolysis of copper sulphate solutions with a rotating copper cathode the potential drop across the cell, with constant current, increases with an increase in the speed of rotation. In view of the fact that a more efficient stirring of the solution is obtained by increasing the rotation, concentration differences being thereby lessened, a drop in potential difference might be expected as a random prediction. Since this increase is rather large, and seemingly abnormal, it was deemed advisable to attempt an explanation of this phenomenon. In the present paper the details of this investigation are given. The results are summed up as follows:

1. There is a definite measurable increase in potential difference in the electrolysis of acid copper sulphate solution, as the speed of rotation of the cathode is increased from 1000 to 5000 r.p.m., for instance.

2. From Fig. 2, the increase indicated to maintain zero current is about 0.010 or 0.015 volt.

3. This can be accounted for by the increase in the rotating electrode potential, with increased rate of rotation.

4. The increase in the potential of the rotating electrode with increased rotation is due to the tendency to form cuprous ions, the increase being greater when conditions are favorable for the formation of cuprous ions, and *vice versa*. As the rotation increases, the stirring becomes more efficient, the cuprous ions are removed, and the solubility of the copper is therefore increased. This gives rise to a higher voltage.

5. The potential of the rotating electrode (cathode when current is flowing) may be much higher than that measured here without current, probably due to the removal of cuprous ions from the solution by precipitation.

6. The increase in the drop of potential with increasing rate of rotation of the cathode in the copper-copper sulphate electrolysis is probably due in part to an increased resistance.

7. The decrease in the potential of the electrode in cuprous chloride solution is due to the more rapid removal of the cupric ions with stirring, thus removing the reagent dissolving the copper.

8. The potential of a rotating electrode in nitric acid goes down with increasing rate of rotation, due to throwing off more and more thoroughly the film of nitrous acid, thus decreasing the solution of the copper, and therefore its voltage.

9. The solution pressure of a metal is increased if the size of its crystals be decreased. This is made apparent by an increase in the potential of the metal as the crystal size is decreased, by hammering or burnishing, for instance. It is also shown by the fact that fine-grained metal (electrolytic copper) is electropositive to more coarsely crystalline metal (cast copper).

The paper was briefly discussed by Drs. Miller, Langmuir and Richards.

Hyperbasis

On Friday evening Prof. **F. B. Kenrick,** of the University of Toronto, gave a most interesting experimental lecture entiled "Hyperbasis"—a word introduced to cover such phenomena as superheating, supercooling, and supersaturation. The experiments, which included several new applications of projection, were shown on the screen with the help of a Bausch & Lomb balopticon.

The demonstrations were brilliant and brought out again the wonderful experimental skill of Professor Kenrick.

Although for the most part the study of hyperbasis has led to somewhat irregular and indefinite results, there are a few empirical rules which may be stated in a general way.

In the first place, it has been found practically impossible to supersaturate a liquid in respect to another liquid to any appreciable extent. In illustration of this a flattened tube containing a solution of phenol and water was shown on the screen alongside a thermometer. These were contained in a water cell the temperature of which was alternately lowered and raised. The liquid in the tube became suddenly opaque and transparent, alternately, as the saturation point was passed.

Secondly, liquids when slightly superheated or slightly supersaturated with a gas produce bubbles only on contact with a gas or a porous substance containing a gas. A tube of ether was shown on the screen and heated to 50 deg. C. without any sign of boiling. A capillary tube of air and a grain of pipeclay caused violent ebullition, while a platinum wire caused only a temporary disturbance.

Thirdly, the crystallization of liquids, slightly supercooled or slightly supersaturated with a solid, is brought about only by contact with the solid itself or with an isomorphous substance. Two experiments illustrated this rule. A dish with $CuCl_2$ previously painted on the bottom with copper chloride solution and dried was shown by vertical projection. The symbol $CuSO_4$ was then scratched on the glass with a crystal of bluestone, and a supersaturated solution of copper sulphate poured into the dish. The $CuCl_2$ disappeared at once, while the invisible $CuSO_4$ was gradually developed by precipitation of bluestone crystals. Next a plate covered with a supercooled solution of m-Chlornitorbenzol in Canada balsam was scratched with the corresponding p-chlor and m-brom compounds; the former had no effect, while every movement of the m-brom compound was followed by a train of crystals.

The next part of the lecture dealt with more extreme cases of hyperbasis in which reaction takes place spontaneously and often in an apparently quite capricious manner. Even here, however, certain irregularities might be observed. The influence of the quantity of substance was shown by gradually heating simultaneously three tubes of ether of different sizes. The contents of the largest tube exploded at about 65 deg.—30 deg. above boiling point. In the second tube the ether did not explode till the temperature had risen many degrees higher, while the smallest tube was heated to 90 deg. without hyperbasis being relieved spontaneously, although the ether could be seen evaporating with great rapidity at the free surface.

This was followed by a number of experiments explanatory of the conditions under which spontaneous crystallization of liquids occur. The relation between temperature and linear rate of crystallization was first shown by allowing melted vanillin to crystallize in a capillary tube. The *temperature* was measured by the projection thermometer, the *distance* by the scale divisions on the same thermometer, while the *time* was indicated by the shadow of a pendulum, beating seconds. At intervals of 20 seconds the temperature of the cell was changed and successive readings were marked on a plate of smoked glass. By the time the crystallization had reached the end of the tube the complete curve had been drawn, showing the initial increase in rate below the melting point, the well-known interval of constant rate, and the final rapid decrease in rate with falling temperature.

The dependence of the number of nuclei on the temperature and duration of supercooling and on the presence of admixtures was shown in a tube of vanillin by Tammann's "exposure and development" method. A double water-cell was used, each half containing a transparent thermometer visible on the screen. The cell was connected to a hot and cold water system by which each half could be rapidly heated or cooled to the desired temperature. In this cell all the operations of melting, exposure, and development could be followed on the screen.

The various forms of vanillin and the relative number of nuclei of each form were shown in the projecting microscope with polarized light. In these experiments also the effect of time and temperature of exposure was illustrated.

The importance of Tammann's work on nuclei was emphasized in connection with the preparation of amorphous substances and of various crystalline forms of the same substance, and also in its bearing on the structure of metals and alloys.

A curious experiment, the result of which the lecturer did not attempt to explain, was shown by adding a dilute magnesia mixture to a dilute solution of sodium phosphate, and then scratching the bottom of the vessel with a rod at intervals of five seconds. The crystals of ammonium magnesium phosphate appeared only where the glass had been scratched fifteen seconds or more after mixing. It was suggested that this might be due either to the slow formation of nuclei or to the time necessary for a very intimate mixing of the liquids.

The lecturer concluded with an experiment on the electrolysis of potassium hydroxide solution with a mercury cathode (Hulett). The surface of the mercury was shown by the reflectoscope while a current of 8 amperes was passing. Although a considerable quantity of potassium was deposited in the mercury no evolution of hydrogen took place at first except when the mercury was touched with a platinum wire. Finally solid crystals of potassium amalgam separated out, and when these covered the surface of the mercury violent effervescence occurred.

Saturday Sessions

The two sessions of Saturday were devoted to the presidential address of Dr. Miller and the symposium of papers on the electrodeposition of metals. The meeting was practically interesting as the National Electroplaters' Association had been formally invited to attend it and discuss the papers, and some twenty or thirty of its most prominent members were present. The meeting, therefore, gave an opportunity for a very interesting exchange of views and ideas between scientific electrochemists, industrial electrorefiners and practical electroplaters.

Presidential Address on Electrode Reactions

In opening the symposium, the president, Dr. **W. Lash Miller,** of the University of Tonorto, expressed the thanks of the Society to those members who had undertaken the laborious task of collecting and arranging the literature of the subject, and had thus rendered possible the discussions of the day. The subject, he said, had been brought before the Society nine years ago at the time of the St. Louis fair in a paper by Dr. W. D. Bancroft on "the chemistry of electroplating," in which the author, among other matters, emphasized the analogy between the structure of metals deposited electrolytically by a heavy current and the structure of precipitates produced rapidly by purely chemical means. In so doing, Dr. Bancroft laid the foundation of the only theory we possess of the direct effect of current density on structure, and was one of the first to preach the modern view that electrochemistry is a branch of chemistry, with all that that implies. Faraday's law and $\log c_1/c_2$ are, of course, as essential as ever, and electrochemical phenomena have not ceased to obey the laws of energy; but it is now recognized that they obey the Ostwald-Bancroft law as well, according to which the thermodynamically "most probable" reaction is in practice perhaps the least likely to occur.

More than any other one subject, the study of the effect of current density on the reactions at the electrode has held the interest of electrochemists during the past decade. In a few instances a change in the nature of the product of electrolysis with rise in current density has been observed, as when

Skrabal showed the analogy between the electrochemical reduction of chromic acid by heavy currents and the ordinary chemical reduction of the same substance by quick-acting chemical reducing agents; but in most of the cases studied, no difference in the final products with different current densities has been noted, and the interest has lain in the endeavor to account for unexpected polarization.

It has long been known, of course, that electrolysis may cause differences of concentration at the two electrodes, and thus build up a polarizating electromotive force; indeed, until it was shown by an application of the theory of diffusion such concentration differences might be calculated, the tendency was to ascribe all polarization to this cause. The diffusion theory as first used, by Weber, Sand, and Warburg, was applicable only to cells in which convection—whether caused by stirring, gas evolution, temperature changes, or concentration differences—was absolutely excluded. This condition is with difficulty complied with, and rapid progress dates only from the application, by Nernst, Brünner, and Merrian, of Noyes and Whitney's theory of a surface film of liquid adhering to the immersed solid, through which the constituents of the solution can reach the electrode by diffusion and migration only.

It is now possible to calculate the concentrations at the surface of a rotating electrode from the current density, diffusion constant, and other easily accessible data, so that in any given case the amount of polarization due to concentration differences may be calculated and compared with that actually observed. In the case of copper deposition from a copper sulphate solution, Sand observed a polarization much greater than could be accounted for by the concentration effect, and the same is now known to hold for many other metals. Sand asscribed the excess to a "resistance," LeBlanc to the time needed for the dehydration of copper ions, Reichenstein to the deposition of a copper-hydrogen compound as primary product of electrolysis, while Bennett, in his paper of the day before, showed that under certain assumptions the effect might be regarded as due to low concentration of cuprous sulphate at the cathode.

Working under Reichenstein's conditions, with slightly acid copper sulphate solution, Mr. Burt-Gerrans and Mr. Brant were able, like him, to reach the high cathodic overvoltage of 0.7 volt. On adding nickel sulphate to the solution, the overvoltage persisted for ten minutes or so, but no trace of nickel was deposited. Similarly, when the electrolyte contained copper and tin, or silver and copper respectively, and the overvoltage was sufficient, on paper, to deposit tin in the first instance and copper in the second, no traces of these metals were in point of fact deposited. Such negative results discredit explanations based on rates of reaction, intermediate alloy, and ion concentration; the more so that in the experiments next to be referred to positive results were forthcoming.

Ever since Smale's determination of the e.m.f. of the oxygen-hydrogen cell was shown by Haber to be 0.2 volt or so below the thermodynamically reversible e.m.f., the "oxide theory" of Nernst, Luther, and Lorenz—that the e.m.f. of an oxidation or reduction cell with "unattackable" electrodes is due to the existence of oxygen or hydrogen compounds of the electrode metal—has been generally accepted. According to this view, the long delay which Luther found to occur before a "chromic acid electrode" reached its stationary e.m.f. must be ascribed to slow oxidation of the platinum by the chromic acid; the reaction between the electrode and iodine, on the other hand, is practically instantaneous.

This theory can be checked by an experiment analogous to that with the copper and nickel, by introducing a cathode into a solution in which chromic acid is slowly being reduced by an iodide. The e.m.f. thermodynamically "necessary" to reduce the chromic acid is less than that needed to reduce the iodine; experiments carried out by students show, however, that, in fact, iodine is reduced. Similarly when the chromic acid is replaced by arsenic acid; in this case the reverse reaction also is accessible, and in solutions where arsenious acid was being oxidized by iodine, an anode was found to bring about the "less probable" oxidation of the iodide.

The most striking case of this type was that studied by Mr. Welsh some years ago, the details of which are unfortunately not yet in print, in which the cathode was found to reduce iodine, not ferric salt, in solutions where all the time the ferric salt was slowly being reduced by an idodide. This case, even more than the others, emphasizes the oversimplicity of the older electrochemical view of these "ion reactions," and of the graphic method by which on the blackboard a ferric salt, Fe^{+++}, is reduced by rubbing out a +.

In closing, Dr. W. Lash Miller referred to the catalysis of these oxidation reactions by platinum and the absence of catalysis by gold, connecting this with the difficulty experienced in determining the e.m.f. of a "chromic acid electrode" in which the metal used was gold. He also spoke of the application of the theory of diffusion to cases where a precipitate is formed at the electrode, illustrating by a very interesting demonstration the periodic phenomena discovered by Küster during the electrolysis of a solution of sodium sulphide.

Symposium on Electroplating

In the symposium on electroplating five papers were presented, all of which had been prepared at the request of the President of the Society, Dr. Miller. These papers were intended to compile all available literature on electroplating, to compile all recipes for plating the different metals and all other information on electroplating that could be found in the literature. This set of papers, therefore, represents a real compendium on electroplating. As these papers were themselves concise abstracts of the available literature, it is clearly impossible to abstract them again, and only the contents of the different papers will be given here. They will at least show the enormous amount of information stored up in these papers. For the full papers the readers must be referred to the next volume of the Transactions of the Society.

Electrodeposition of **Gold** and **Silver**. By Dr. **Francis C. Frary,** University of Minnesota (73 pages). Part I: Electrodeposition of gold: simple halide baths, cyanide baths, gold chloride in potassium cyanide, gold cyanide in potassium cyanide, fulminating gold in potassium cyanide, miscellaneous cyanide baths containing little or no cyanide; baths to produce special colors, general information. Part II: Electrodeposition of silver: Simple salts, cyanide baths, silver nitrate in potassium cyanide, silver chloride and potassium cyanide, silver cyanide in potassium cyanide, potassium cyanide and miscellaneous salts, baths for bright deposits, general information on cyanide baths, ferrocyanide baths, miscellaneous chloride mixtures, miscellaneous non-cyanide solutions, alloyed deposits, general information on silver plating.

Electrodeposition of **Nickel** and **Cobalt.** By Dr. **Oliver P. Watts,** University of Wisconsin (53 pages). Electrodeposition of cobalt, fifty-one solutions for electroplating with cobalt, deposition of an alloy of cobalt and nickel, electroanalysis (fourteen solutions for estimation of cobalt and nine solutions for separation of cobalt and nickel). Electrodeposition of nickel, purity of nickel anodes, passivity of nickel anodes. Nickel-plating baths: Baths containing single salts of nickel (fourteen neutral, three acid, and nine alkaline baths); baths containing double salts (twelve neutral and eighteen acid baths); thirty miscellaneous baths; malleable nickel (three solutions); three baths producing thick deposits; addition agents for the production of smooth deposits; nickel electrotypes; the nickeling of zinc; electrolytic determination of nickel.

Electrodeposition of **Copper.** By Dr. **C. W. Bennett,** Cornell University (18 pages). Eight acid solutions; twenty-three alkaline solutions; addition agents.

Electrodeposition of **Brass** and **Bronze.** By Dr. **C. W. Bennett,** of Cornell University (10 pages). Twenty brass solutions; eleven bronze solutions.

Electrodeposition of **Lead.** By Dr. **Frank C. Mathers,** of Indiana University (38 pages). Lead salts of inorganic acids (fluorine acids and perchloric acid); lead salts of nitrogen acids for the deposition of metallic lead; salts of other in-

organic acids; lead salts of acetic, oxalic and other organic acids; alkaline solutions; lead peroxide deposition; non-aqueous solutions; miscellaneous; electroplating and coloring with lead.

Electrodeposition of **Tin.** By Dr. **Edward F. Kern,** of Columbia University (25 pages). Electrodeposition of tin by immersion; electrolytic determination of tin, with reference to the use of rotating electrodes and the removal of tin deposits from the cathode; electrodeposition of tin by separate current (nineteen solutions); conditions necessary to obtain bright, dense, adherent deposits of tin; electrolytic refining of tin; effect of organic addition agents in tin electrolysis (sodium-stannous chloride electrolyte, stannous fluoride electrolyte, sodium-stannous fluoride electrolyte).

The president, Dr. Miller, welcomed courteously the members of the National Electroplaters' Association, which had come to join in the discussion. For the National Electroplaters' Association hearty replies were made by Mr. Proctor, founder of association, and Mr. Hogaboom, secretary of the association.

Dr. Wilder D. Bancroft presented the following set of **axioms of electroplating.**

1. Bad deposits are due to excessive admixture of some compound or to excessively large crystals.

2. Excessive admixture of any compound can be eliminated by changing the conditions so that the compound cannot precipitate.

3. Increasing the current density, increasing the potential difference at the cathode, or lowering the temperature decreases the size of the crystals.

4. The crystal size is decreased when there are present, at the cathode surface, substances which are adsorbed by the deposited metal.

5. If a given solution will give a good deposit at any current density, it will give a good deposit at any higher current density, provided the conditions at the cathode surface are kept constant.

6. Treeing is facilitated by a high potential drop through the solution and by conditions favorable to the formation of large crystals.

The discussion was long and at all times animated, interesting, and suggestive. On the side of the American Electrochemical Society the chief speakers, besides Dr. Bancroft, were Dr. Miller, Dr. Bennett, Dr. Richards, Dr. Keith, Mr. Cowles, Mr. Addicks and Dr. Cushman; the chief speakers for the electroplaters were: Mr. Hogaboom and Mr. Proctor; a middle ground was taken by Dr. Kern and Mr. Brown.

It was pointed out by several speakers that the two sides don't yet understand each other. As Mr. Hogaboom said, practical electroplaters don't understand many of the scientific expressions, while scientific electrochemists don't understand the practical conditions under which a plater works. The conditions in the plating shop are very different from those in the laboratory. In the plating shop the electrodes are fixed; there is no physical possibility to regulate the distance between anode and cathode. Then there are perhaps 50 jobs in one big tank and a heavy deposit must be obtained of exactly the right color in a short time. For instance, when bar bottles are to be silver-plated, forty-eight bottles are put in at the same time and they are to be taken out at the same time. It is easy enough in the laboratory to measure and adjust the current density. But in the plating shop that would mean first to answer questions like this: How many square inches is the surface of a gross of table forks? There is a great thirst for knowledge among electroplaters—thirst for chemical knowledge.

Mr. Lidbury suggested that electroplaters should give up the use of terms like ounces per gallon, degrees Baume, and the like and go over to the metric system. Dr. Miller praised Hering's excellent Conversion Tables and recommended them to the Electroplaters' Association, while Mr. Hogaboom suggested that conversion tables may also be used backwards.

As Mr. Hogaboom pointed out, the fundamental unit of the electroplater is the volt. He talks volts because his experience shows that he gets the desired result if he uses so and so many volts.

Dr. Kern pointed out that the difference betwen the ways of electroplaters and electrochemists is more apparent than real. He showed in detail that while one specifies volts and the other current densities, both are in fact doing the same thing.

Mr. Addicks said that the ordinary plating solutions should be adjusted periodically and brought back to standard as is done in electrolytic copper refineries.

Dr. Kern remarked that if it is said that a solution in a plating shop is 18 years old, this must not be taken literally. Water is added from time to time. Salts are added. The plater really maintains a fair degree of purity. His control is by adjusting the degrees Baume.

Mr. Addicks said that platers should not insist so strongly on absolutely pure salts. For the sake of fair play they should permit the salts to contain a reasonable degree of impurities just as the water and the anodes are not absolutely pure.

Mr. P. S. Brown recommended to encourage the manufacturer to send samples to chemists if he does not want to employ chemists himself. Of greatest importance is the organization of a plating shop on the basis of scientific management—"Taylorizing" a plating shop. If this is done, it can always be shown why a thing pays. It could be easily shown in this way why chemical control pays in a plating shop. The old antagonism of the plater against the chemist is now vanishing and things have been changed greatly.

Dr Richards remarked that it is often said that the Germans produce things so cheaply. The reason why this is so is that no German plant is without a chemist. The chemist reduces the total cost instead of adding to the expense.

With respect to **gold plating,** Mr. Hogaboom pointed out that a gold deposit reflects the color of the surface on which it is deposited. The reflection of the light from the surface under consideration determines the solution to be used. There is a great difference between inside and outside color if the same solution is employed for inside and outside gold-plating. Outside gold is translucent. If the outside is first plated with copper and the gold is then plated on it, it is possible to get the same color outside and inside. Mr. Hogaboom also referred to additions of arsenic or lead in the deposition of gold and said that it is a disputed question whether arsenic or lead is better.

The importance of getting a right color was also emphasized in connection with **bronze plating.** When the chemist speaks of bronze plating he means deposit an alloy of copper and tin. When the electroplater speaks of bronze plating he means getting a color that is the true color of bronze. Some of the so-called bronze-plating solutions contain copper and zinc, a few others copper and nickel. The much greater cost of tin as compared with zinc is, of course, an item of commercial importance.

With respect to **copper plating** Mr. Hogaboom said that acid solutions are better than alkaline solutions from the electroplater's standpoint and that all heavy copper deposits require an acid solution. The only one satisfactory addition agent for acid sulphate solution for copper plating is, according to Mr. Hogaboom, common alum.

It was to **nickel plating** that quite a considerable part of the discussion referred during the day. Mr. Hogaboom said that the necessity of copper plating as a preliminary basis for nickel plating on iron, steel, etc., was questionable and referred to the success of direct nickel plating on hammer heads. He also emphasized the enormous amount of damage which a small amount of copper can do in a nickel solution.

Mr. Brown said that there was a reason for copper plating before nickel plating. It has essentially a cleansing effect and yields, therefore, better adherence and better deposits.

Mr. Hogaboom emphasized the commercial difficulties in nickel plating by stating that nickel plating of the common three-hole gas plate must be done for 23 cents and that this has to pay for labor, nickel, and everything.

Dr. Bennett thought that pure electrolytic nickel anodes would be better for nickel plating than the ordinary nickel anodes, which always contain iron.

Mr. Hogaboom replied that one of the largest manufacturers of nickel anodes in this country bought a great many tons of electrolytic nickel and could not melt it and cast it into anodes by the ordinary methods.

Dr. Bancroft thought that this fact suggested the advantages of trying something on a laboratory scale before trying it on a ton scale.

Dr. Richards thought that manufacturers of electrolytic nickel might be induced to produce nickel anodes directly.

Mr. Hansen said that electrolytic nickel could be successfully melted in an electric arc furnace and cast with homoeopathic quantities of silicon and aluminium and could be forged, and that they had done it with 1000-lb. batches.

With respect to **lead plating** Mr. Proctor said that electroplaters were interested in it only when they were called upon to produce "royal copper" and metallochromes (iridescent surfaces).

Dr. Keith spoke of his early work with electrodeposition of lead.

With respect to **tin** Dr. Keith gave interesting reminiscences of electrolytic detinning in very early days.

Dr. Cushman spoke at some length on the determination of tin in foodstuffs contained in tin cans. This is a matter which is of immense importance at present from a commercial viewpoint.

With respect to **addition agents** Mr. Addicks referred to the glue in the barrels used in the early days of nickel plating and called attention to the almost universal use of adding some common salt to the electrolytic copper refining solution, though the reason for this is not known. He also emphasized that electrolytic refining, electrotyping and electroplating are very different in character and scope.

With reference to addition agents and metallurgical flotation processes a pretty demonstration was made by Professor Bancroft. A complete account of the discussion will be found in the next volume of the *Transactions* of the Society.

Committee Appointments.

Dr. Baskerville spoke briefly of the prevention of occupational diseases in the chemical industries and made a motion that a committee of the American Electrochemical Society be appointed to act together with a similar committee of the American Chemical Society. The motion was carried unanimously, and President Miller appointed Mr. F. A. Lidbury, chairman of the committee, together with Mr. A. H. Hooker and another member still to be appointed.

In order to continue the co-operation between the American Electrochemical Society and the National Electroplaters' Association a committee was appointed to summarize what has been accomplished so far and especially to work out simple standard methods of analyzing plating baths, to work out standard plating baths, and to account for many of the puzzling phenomena noted by practical platers.

Dr. W. D. Bancroft is the chairman of this committee, and Mr. Hogaboom and Mr. Proctor are the representatives of the Electroplaters' Association on the committee, while the committee has power to add further members of the American Electrochemical Society.

In the following we give a complete alphabetical list of the names of members and guests who registered at the Atlantic City meeting:

Paul O. Abbe, New York; Lawrence Addicks, Chrome, N. J.; Thos. B. Allen, Niagara Falls, N. Y.; Richard Amberg, Harrison, N. J.; G. Aminoff, Baltimore, Md.; W. C. Arsem, Schenectady, N. Y.; W. Asef, Philadelphia, Pa.; Chas. W. Bailer, Philadelphia, Pa.; Mr. and Mrs. T. F. Baily, Alliance, O.; Wilder D. Bancroft, Ithaca, N. Y.; Samuel Barr, Philadelphia, Pa.; Chas. Baskerville, New York; E. A. Beck, New York; E. H. Bedell, New York City; C. W. Bennett, Ithaca, N. Y.; R. Beutner, New York; R. S. Bicknell, New York City; H. Bloodsworth, Lansdowne, Pa.; Earl Blough, Pittsburgh, Pa.; J. E. Breckenridge, Woodbridge, N. J.; P. S. Brown, New York City; de Comey B. Browne, New York; Chas. L. Bryden, Scranton, Pa.; I. H. Byrne, Cleveland, Ohio; W. R. Clymer, Cleveland, Ohio; Mr. and Mrs. A. M. Comey, Chester, Pa.; W. A. Cowan, New York; Mr. and Mrs. Alfred S. Cowles, Sewaren, N. J.; Edwin L. Crosby, Detroit, Mich.; T. E. Crossmay, New York City; C. R. H. Cunningham, Lansdowne, Pa.; Allerton S. Cushman, Washington, D. C.; Newton E. Dabolt, Pittsfield, Mass.; Homer T. Darlington, Natrona, Pa.; R. W. Davis, Jr., Aspinwall, Pa.; C. Dittmar, New York City; Earl Eastman, Atlantic City, N. J.; Mr. and Mrs. Francis A. J. Fitzgerald, Niagara Falls, N. Y.; Harry C. Flanigan, New York City; K. G. Frank, New York City; Milton W. Franklin, Schenectady, N. Y.; A. E. Gibbs, Philadelphia, Pa.; Carl H. Gille, Philadelphia, Pa.; H. W. Gillet, New York; J. S. Goldbaum, Norwood, Pa.; S. L. Goodale, Pittsburgh, Pa.; Mr. and Mrs. Thos. D. Greenley, Montclair, N. J.; C. A. Hansen, Schenectady, N. Y.; August Heit, Philadelphia, Pa.; H. Hennauer, Philadelphia, Pa.; Carl Hering, Philadelphia, Pa.; A. T. Hinckley, Niagara Falls, N. Y.; Alcan Hirsch, New York City; George B. Hogaboom, Newark, N. J.; L. S. Holdstein, Palmerton, Pa.; Mr. and Mrs. A. H. Hooker, Niagara Falls, N. Y.; O. T. Hungerford, Belleville, N. J.; W. R. Ingalls, New York City; Mr. and Mrs. Alois von Isakovics, Monticello, N. Y.; H. T. Kalmus, Kingston, Ont., Can.; N. S. Keith, Philadelphia, Pa.; H. W. Kellogg, Niagara Falls, N. Y.; Philo Kemery, Pittsburgh, Pa.; Frank B. Kenrick, Toronto, Ont., Can.; Edward F. Kern, New York City; A. S. Knight, Niagara Falls, N. Y.; J. Ludwig Koethen, Jr., Atlantic City, N. J.; Frederick L. Koethen, Niagara Falls, N. Y.; W. F. Kutley, East Orange, N. J.; Mr. and Mrs. Irving Langmuir, Schenectady, N. Y.; F. A. Lidbury, Niagara Falls, N. Y.; Chas. F. Lindsay, Bridgeport, Ct.; Stewart J. Lloyd, University, Ala.; B. F. Lovelace, Baltimore, Md.; D. A. Lyon, Pittsburgh, Pa.; G. M. J. Mackay, Schenectady, N. Y.; Wallace W. Mann, Cleveland, Ohio; C. W. Marsh, Niagara Falls, N. Y.; G. Stanley Merkle, Schenectady, N. Y.; W. Lash Miller, Toronto, Ont., Can.; Mr. and Mrs. Harlan S. and Miss L. C. Miner, Gloucester, N. J.; Walter S. Moody, Pittsfield, Mass.; Mr. J. M. and Miss Helen Muir, New York City; Mr. and Mrs. E. F. Northrup, Princeton, N. J.; J. T. Owens, Newark, N. J.; B. J. Owens, Newark, N. J.; Chas. L. Parsons, Washington, D. C.; N. Petinot, Niagara Falls, N. Y.; William T. Price, Philadelphia, Pa.; Chas. H. Proctor, Arlington, N. J.; Dr. and Mrs. Jos. W. Richards, South Bethlehem, Pa.; E. D. Rippel, Buffalo, N. Y.; E. F. Roeber, New York City; G. A. Roush, South Bethlehem, Pa.; S. S. Sadtler, Philadelphia, Pa.; Pedro G. Salom, Philadelphia, Pa.; L. E. Saunders, Niagara Falls, N. Y.; Carl G. Schluederberg, Pittsburgh, Pa.; Wm. J. Schneider, New York City; Job. Schoenawa, Menzenschwand, Germany; Frederick F. Schuetz, Newark, N. J.; Mrs. J. S. Scott, Philadelphia, Pa.; J. A. Seede, Schenectady, N. Y.; Philip Sievering, Newark, N. J.; Dyer Smith, New York City; Mrs. G. D. Smith, Montclair, N. J.; Harold H. Smith, Orange, N. J.; H. H. Smith, Newark, N. J.; Mr. and Mrs. Acheson Smith, Niagara Falls, N. Y.; F. D. Steik, Passaic, N. J.; J. E. Sterling, New York; Chas. A. Stiehel, Newark, N. J.; Walter T. Taggart, Philadelphia, Pa.; Mr. and Mrs. S. F. Tombaugh, Alliance, Ohio; H. M. Turner, Jr., Pittsburgh, Pa.; C. H. Vom Baur, New York; L. D. Voon, Detroit, Mich.; M. G. Weber, Perth Amboy, N. J.; Chas. A. Weeks, Philadelphia, Pa.; R. J. Weitlaner, Philadelphia, Pa.; Arthur B. Wells, Philadelphia, Pa.; R. H. White, Niagara Falls, N. Y.; Mr. and Mrs. A. S. White, Philadelphia, Pa.; W. R. Whitney, Schenectady, N. Y.; F. C. Woodside, Pittsfield, Mass.; F. Zimmermann, Newark, N. J.

Coming Meetings.

The next meeting of the American Electrochemical Society will be held in Denver, Col., early in September, and the principal feature of this meeting will be a symposium of papers on what electrochemistry can do for Western metallurgy.

The spring meeting of 1914 will be held in New York City.

For the autumn meeting of 1914 the society has received a courteous invitation to go to Birmingham, Ala., and see the New South.

In 1915 one of the meetings will be held in San Francisco in connection with the world's fair.

Dr. Joseph W. Richards, Lehigh University, South Bethlehem, Pa., is secretary of the society.

Filter Press.—The Berrigan filter press has been taken over by the Hy. R. Worthington Pump Company, at Harrison, N. J.

British Iron and Steel Institute.—The annual meeting of the (British) Iron and Steel Institute will be held in London on May 1 and 2. The Bessemer gold medal will be awarded to Mr. Adolph Greiner, general director of the Société Cockerill, Seraing. The autumn meeting will be held at Brussels.

The lead production of the United States for 1912 has been finally announced by the Geological Survey. The total production of refined lead was 480,894 short tons, which is 71 tons less than the estimate given out on Jan. 2, 1913. This is exclusive of 13,552 tons of antimonial lead. The total refined lead output was less than that of 1911 by 6085 tons, a decrease of 1.2 per cent. Missouri had the greatest output, 162,610 tons, a decrease of about 20,000 from the previous year. Idaho was next with 127,707 tons, a gain of about 10,000 tons over 1911.

The iron ore deposits at Minaret, Madera County, California, are said to be among the largest in California and perhaps in the West. The next in size, but of greater commercial importance, are the Eagle Mountain deposits, in Riverside County, California. The latter have been examined by the Geological Survey and reported in *Bulletin* 503. A moderate estimate of the total quantity of mixed ore and gangue material available in the deposits now exposed, is about 75,000,000 tons. About four-fifths of this is assumed to be ore of very high grade. The principal obstacle to the iron industry in California is the absence of fuel. The future operation of the Alaska coal fields is expected to stimulate iron reduction and steel refining in California.

Notes on Chemistry and Metallurgy in Great Britain

From our Special Correspondent

The Institute of Metals

The annual general meeting of the Institute of Metals was held at the Institution of Mechanical Engineers on March 11 and 12, 1913. In the course of his presidential address, Professor **A. K. Huntington** said it was usual in a presidential address to call attention to the lines on which the Institute could be most advantageously developed, and a study of the purposes and progress of other similar institutions of longer standing would afford assistance in this direction.

In 1894 in his presidential address to the Institution of Mining and Metallurgy he said "the progress in the iron and steel industries in the last twenty-five years has been extraordinary, and I have no hesitation in saying that a large share of it is due to the existence of the Iron and Steel Institute. The members of those industries are no longer working against one another in a narrow spirit, afraid to confide their ideas to one another lest they should give a competitor an advantage. All this narrowness has disappeared, and now members visit one another's works and discuss everything concerning them at every opportunity. As in everything else the best man wins in the long run, while the rest of the world is benefited by more rapid progress."

At the time that address was made, he had in mind many works in this country where metals were smelted and refined which preserved secrecy as to their doings, and would not admit anyone to their works. Now, notwithstanding that the Institution of Mining and Metallurgy had been in existence twenty-one years, these same works, for the most part, maintain that same attitude: they contribute nothing whatever to that institution. That institution had had its time fully taken up with metallurgy as carried out in direct connection with mines, and probably had hardly given the works in question a thought; and judging from the experience of the Institute of Metals, it would probably have only been rebuffed had it acted otherwise. He did not think the Institution of Mining and Metallurgy would object to such works coming under the Institute of Metals. Perhaps they would come into line in time, but that time was not yet.

The way in which the Institute of Metals had progressed during the few years of its existence left no doubt, to his mind, that the non-ferrous industries would respond to its stimulus just as the iron and steel industries had done in the case of the Iron and Steel Institute. There was no doubt, too, that the Society of Chemical Industry had done good work in the field covered by the Institute of Metals; and they should be grateful to that society, which would be the first to admit that the time was now ripe for specializing more fully in this subject.

It must be remembered that metallography for practical purposes had only come into existence within about the last fifteen years, and the difference it had made in explaining what occurred to metals and alloys in the course of manufacture and in use was incalculable.

It was easy now to appreciate the fact that the knowledge to be gained from chemical analysis of an alloy was as nothing compared with that which could be obtained by studying the internal condition and structure of metals and alloys when their composition was varied, and they were subjected to suitable or unsuitable mechanical and heat treatment.

In dealing with copper-zinc alloys in the alpha-beta range it had been the practice in his laboratory for several years to employ a method of micrographic mensuration which occupied quite a short time compared with the chemical analysis formerly required.

This was only the fifth year of the existence of the Institute, and the membership was 606. At the same period of its existence the Iron and Steel Institute had 587 members, and the Institution of Mining and Metallurgy reached practically the same figure, 585, in its seventh year.

The position between the manufacturer and the user was somewhat delicate. If the manufacturer inquired of the user as to the behavior of the material he had supplied, the user might turn round and say, "Dear me! What is the meaning of this? Evidently he has no confidence in the material himself after all he has said about it." So it came about that the manufacturer waited till the user went for him, and in the very human desire to save his own skin he probably retorted that what had gone wrong was owing to improper usage; and in ninety-nine cases out of a hundred he was most likely right.

What was required was more mutual confidence, and when users realized that other men were quite as straightforward as themselves a great advance would be achieved. It may be taken as certain that manufacturers were in general straightforwardly doing their very best to acquire or maintain a good reputation. His personal experience was that the greatest difficulty he had had to contend with was that of obtaining reliable information from users, who very frequently had not got it.

Accurate observation demanded requisite training from very early youth upwards, and now that metallography and physical chemistry were taught in so many institutions young engineers and works chemists should be required to have competent knowledge of those subjects.

The institution was very often in the position to bring the manufacturer and the user into sounder relations by pointing out that users had everything to gain by carefully and systematically observing what was going on with the metals and alloys they were employing, and by giving friendly information to the manufacturer which would assist him in meeting their requirements. In this way the users would also learn a great deal and would realize that the old saying "Prevention is better than cure," most certainly led to economy and peace of mind.

Papers and Discussions

Abstracts of all the papers presented have already been given in our April issue, page 179. In the following we give a report of the discussion:

Corrosion of Distilling Condenser Tubes

A paper entitled "Contributions to the History of Corrosion (Part 2), having special reference to the Corrosion of Distilling Condenser Tubes" was presented by Mr. **Arnold Philips,** the Admiralty chemist. (See abstract on page 179 of April issue.)

In the discussion, Dr. Bengough said he thought that the author had proved his case, and that the hydrochloric acid was the cause of corrosion. Experience in connection with brass led him to accept this view. Corrosion of this description was more prevalent in the Merecantile Marine than in the Navy, probably owing to the more careless treatment of condensers in the Mercantile Marine.

Sir Gerard Muntz said that not very long ago it was the general impression that the Admiralty never had any trouble with their condenser tubes, and had attained a state of perfection. The paper gave point to a fact which he had been trying to drill into the heads of people for many years—namely, that the cause of trouble must not always be sought where the trouble is found, and it did not follow that if the cause of corrosion took place in a condenser tube that the trouble arose there. He had recently had the opinions of five distinguished scientists to the effect that they had never found corrosion to arise from anything due to the constitution of the tube.

Dr. G. H. Bailey did not accept the view that hydrochloric acid was necessarily the cause of the corrosion mentioned by the author. It always seemed to him that the dissolved oxygen in the water was an important factor, because when aluminium was exposed to the action of water which had been boiled until all the air was expelled, there was no action. The same thing occurred even with a strong salt solution when air was absent. Did the author say he was not able to determine the presence of hydrochloric acid because it was volatile? His own

experience was that the acid water could be concentrated by boiling so as to enable the amount of hydrochloric acid to be determined.

Mr. Rhead asked whether the author had ascertained that hydrochloric acid would attack copper in the absence of oxygen. The paper did not afford sufficient evidence to combat the accepted idea that copper was not directly attacked by hydrochloric acid, but that in the presence of oxygen it was acted on freely.

Mr. Phillip, in his reply, said that throughout the paper when referring to hydrochloric acid he had meant hydrochloric acid admixed with air. The evidence that the effects had been caused by hydrochloric acid and not by oxygen alone was quite clear because with totally submerged steam heating coils no corrosive effect was produced comparable with that which took place when the tubes were partly above the surface of the salt water. He believed that the greater cause of 80 to 90 per cent of corrosion occurring in the Navy was the presence of acids, together with electrolytic action. If the tubes were in metallic contact with the steel casing that would account for 99.9 per cent of the corrosion which he had seen.

The president remarked that some time ago a boiler stay of similar composition to those mentioned in the paper was sent to him. In places the zinc was removed and he found that the depth to which the zinc had been removed was in direct ratio of the thickness of the deposit, and at the time he came to the conclusion that the action was due to magnesium salts in the deposit.

The Corrosion of Aluminium

Dr. **G. H. Bailey** read a paper on "The Corrosion of Aluminium," in which he dealt mainly with the effect of water and solution of common salt. (See abstract on page 179 of April issue.)

Dr. Bengough opened the discussion and said that many of the factors affecting the corrosion of aluminium also affected brass, with the difference that in the case of aluminium the oxides were non-adhering, while with brass they adhered firmly. In some respects there seemed to be less difficulty with brass than with aluminium, but on the other hand, secondary reactions occurred necessitating investigation; yet there was not the same difficulty in determining the loss of weight caused by the formation of films of oxide. The idea of direct oxidation was anathema in certain quarters in relation to electrochemical studies; but Dr. Bailey had definitely come to the conclusion that it was an important point in connection with corrosion, and he himself (the speaker) had been gradually coming round to the same view with regard to brass, so that in certain respects he and Dr. Bailey were standing shoulder to shoulder against a great body of physico-chemical opinion.

Dr. Seligman said he did not know a more unscientific method than determining corrosion on the basis of weight. Four of Dr. Bailey's conclusions had already been arrived at by other workers, and he might have referred to those workers more explicitly. He considered that the amount of metal removed was quite unimportant, and, in support of this, produced samples of aluminium corroded by nitric acid and water respectively. He said that estimating the corrosion by the quantity of metal removed they would both be condemned, but the sheet of metal was still in a uniform condition, while the tube was decidedly damaged.

Mr. S. L. Archbutt was of opinion that the times of corrosion adopted by the author were too short. The experience at the National Physical Laboratory was that the formation of a protective coating might exercise considerable influence on the rate of corrosion, which could only be accurately arrived at by much longer periods of immersion than the author had employed. He asked the author how the specimens were immersed, because he had found that whatever supporting materials were used, corrosion was accentuated where the alloy was in contact with them, while friction might also be an important matter.

Mr. Arnold Phillip gave some figures showing that friction had very great influence on the rate of corrosion.

Dr. F. J. Brislee was inclined on the whole to agree with Dr. Bailey's conclusion that oxygen is an important factor in bringing about corrosion. The protective coating was also an important matter. In one case the material had been corroded gradually for three years exposed to spray and only in indirect contact with liquids, but it had subsequently remained practically unaffected. The importance of the protective coating was so fully recognized that patents were being taken out for using it to insulate electrical conductors, and he was astonished to see that a coating of aluminium oxide was flexible enough to permit of the winding of a wire of 18 to 24 gauge on a spool little more than an inch in diameter.

Mr. E. F. Law asked for more data, for he did not think that the results quoted in the paper justified the author's conclusions, although those might still be correct.

Dr. Bailey replied and objected to having been accused of quoting other workers' conclusions without acknowledgment. He had questioned those conclusions, and had therefore worked independently on them and had arrived at the same result. He would not maintain that the amounts of metal removed were the final test, but commercially it was of the greatest importance. He did not think the time of exposure was too short, because longer exposure would mean that the aluminium would be protected by air being driven out of the water.

Engineering Imports and Exports

The returns issued by the Board of Trade for January and February show satisfactory increases as compared with the corresponding period of last year, in respect of all the chief engineering materials except in imports of electrical goods, which are lower. The imports of iron and steel, including manufactures, were £2,683,229, an increase of £661,697; and exports touched £8,866,659, an increase of £959,222. In other metals including manufactures, imports are valued at £5,481,881, an increase of £443,890; and exports are put at £2,421,088, an increase of £543,816. Imports of electrical goods totalled £257,149, a decrease of £17,842; but exports went to £724,583, and increased by £196,503. The imports of machinery reached £1,193,950, an improvement of £169,434; and exports rose to £5,825,103, an increase of £811,805. Imports of new ships, value £2,012, show an increase of £1,505; and exports amounted to £1,123,421, a rise of £411,707. Taking the figures for February alone, the imports and exports of iron and steel show increases of £261,938 and £485,731 respectively; imports of other metals rose £259,360, and exports went up by £228,642; electrical goods exhibited a decline of £14,038 in imports, but exports were £102,976 better; imports of machinery rose £10,654, and the exports increased by £365,973; and new ships showed a rise of £259 in imports, while exports were £104,504 higher.

Market Prices, March, 1913

Copper has been fairly steady over the month, weakening a little about the 6th, but hardening over the last ten days. Opening at £64.17.6, it closes at £67.16.3.

Tin opened at £218, it was at first inclined to go higher, but weakened to £210 on the 13th, since recovering to £219½ and closing at £217.10.0.

Lead opened at £16.12.6 but fell away after the 5th, touching £16.2.6, on the 7th. It has since been stronger, but not above the closing price, £16.15.0.

Haematite opened at 80/6, was 79/- on the 4th, but had reached 82/- by the 15th. It had declined to 79/6 by the 28th and now closes 78/9.

Scotch Pig opened 66/7½ and shows the highest increase of the iron warrants. It rose to 70/- in the first week, then declined a little but recovered by settlement and closes at 72/4½.

Cleveland has improved fairly steadily throughout the month starting at 61/- gaining principally in the second week, it closes at 66/9½.

	£ s. d.
Aluminium, ton	90. 0. 0
Alum, lump, loose, per ton	5.15. 0
Antimony, black sulphide powder, per ton	20. 0. 0

Borax, refined, cwt.	17/6 to 18. 6
Copper ore, 10 to 25% per unit	11/4½ to 11.10½
Copper sulphate, per ton	23.12. 6
Carbolic acid, liquid 97 to 99%, gal.	3
Caustic soda, ash, 48% ordinary, per ton	5.10.0
Ebonite rod, lb.	5. 3
Hydrochloric acid, cwt.	5. 0
India rubber, Para, fine, lb.	3. 8
Mica, in original cases, medium, lb.	3/6 to 6. 0
Petroleum, Russian spot	9¼
Sal Ammoniac, lump, 1st. del. U. K. per ton	44. 0. 0
Sulphate of ammonia, f. o. b. Liverpool	14. 0. 0
Sulphur, recovered, per ton	5. 5. 0
Shellac, cwt., standard orange spot	4. 4. 0
Platinum, oz.	9. 5. 0
Tin ore, 70%, per ton	135 to 137. 0. 0
Zinc, Veille Montagne, f. o. b. Antwerp	28.10. 0

Higher.	£ s. d.	*Lower.*	£ s. d.
Aluminium, ton	1. 0.0	India rubber, lb.	4
Alum, ton	5.0	Sulphate of ammonia, ton	5.0
Copper, ton	2.18.9	Haematite	1.9
Copper sulphate, gal.	1		
Scotch Pig, ton	5.9		
Cleveland	5.4½		

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Concentration of Iron Ores.—In the issue of this journal for December, 1912, page 811, we presented an outline of a paper on this subject, read by Mr. N. V. Hansell, at the Cleveland meeting of the American Institute of Mining Engineers. In the *Bulletin* of the Institute for March, 1913, Mr. C. Q. PAYNE contributes a discussion of Mr. Hansell's paper, from which we take the following extract.

"The high grade of concentrates now produced from magnetic iron ores is particularly striking, as it is not uncommon to find concentrates containing from 67 per cent to 69 per cent iron, representing a recovery of 90 per cent and even 95 per cent. The type of magnetic field producing these results, both in the Gröndal (wet) and Ball-Norton (dry) systems, is one of alternating north and south poles, which was developed by Clinton M. Ball and Sheldon Norton. In the case of wet treatment, the improvement in grade of concentrates is due also to a well-devised scheme of mill treatment, whereby a rough concentrate is first obtained, then reground and reseparated."

In regard to the suggestion of Mr. Hansell to apply the concentrated or condensed-field type of magnetic separator to the wet treatment of feebly magnetic ores, like hematite and limonite, Mr. Payne believes that the suggestion opens an interesting field of inquiry. Mr. Payne did considerable work along this line some years ago, and gives some of the conclusions reached.

"In the first place, hematite particles cannot be so strongly magnetized in a condensed field as to inductively magnetize other hematite particles in their neighborhood, as is the case with magnetite particles. Theoretically, therefore, the sheet of ore fed into the field should be only one particle thick. This means that only a limited capacity may be expected from a relatively expensive machine. Moreover, those ores which have iron-bearing silicates among their gangue minerals, or red colored clays, are difficult to enrich, since for practical purposes such silicates or clays are equally as magnetic in a condensed field as hematite and limonite ore particles, and they are therefore carried with them into the concentrates."

In spite of these obstacles of small capacity and imperfect results, it is unwise to condemn the use of the condensed-field type of electromagnet for the treatment of hematite and limonite ores, either dry or wet, in advance of a careful study of the ore-occurrence in each instance. Mr. Payne inclines to the opinion that a combination of gravity and magnetic separation will give the best results, commercial and technical, in the enrichment of many low-grade hematite ores which cannot be successfully treated by either system alone. Gravity treatment will give better results on coarse sizes; but on the other hand, this system results in high losses in the fine sizes and slime of such ore. It is in the treatment of the latter that magnetic separation accomplishes its best work. These facts suggest the combination of the two methods, each working on the most appropriate material. In such a combination treatment, the finely crushed middlings or tailings should be given a reducing roast to magnetize the iron ore particles, thereby making magnetic separation more efficient. The concentrates thus produced would be of high grade, and the loss would be low. The degree of concentration would be higher by the combination method than by gravity separation alone.

By way of illustrating the possibilities of the combination treatment, Mr. Payne summarizes the results from six Gröndal plants, and the corresponding results from the gravity separation plant of the Oliver Iron Mining Company.[1]

	Crude % Fe	Concentrates % Fe	Tailings % Fe	Recovery %
Average of six magnetic magnetite-ore plants	35.2	67.9	6.2	90.1
One gravity Mesabi hematite-ore plant	35.0	58.0	14.1	80.0

Assuming that the less favorable results of the hematite-ore plant are largely due to the difficulties of treating fine material; also by way of illustration, that magnetic separation applied to this portion would raise the grade of the combined concentrates to 60 per cent iron, then it would be possible to secure 1 ton of such concentrates from 1.96 tons of crude ore instead of 1 ton of such concentrates from 2.1 tons of crude ore. The recovery would thus be increased to 87.4 per cent. Such assumed results would show an increased yield of 1191 tons of concentrates from the 35,000 tons of crude material treated in the mill per day; and taking $2 per ton as the value of concentrates at the mill, would provide an increase of nearly $2500 per day in the gross value of the concentrates from the same amount of crude material.

"The importance of these results will be realized when it is recalled that those iron-bearing deposits of the Lake Superior region alone which contain 35 per cent and more of metallic iron have been estimated to contain 67,640,000,000 tons."

Corrosion of Iron and Overvoltage

The meeting of the (British) Faraday Society, held on April 4, 1913, in Manchester, was devoted to a general discussion of the corrosion of iron and steel.

A paper by Mr. **Bertram Lambert** on an electrolytic theory of the corrosion of iron was first presented. This is published in full elsewhere in this issue.

Prof. **W. W. Haldane Gee**, of Manchester, delivered a lecture, illustrated by experiments and lantern slides, on "The Electrolytic Methods for Preventing the Corrosion of Metals."

He showed that the corrosion of metals can be lessened or prevented in two ways: (1) By connecting the metal to be protected to a more electropositive metal, so that a primary cell is produced; (2) by making the metal to be protected the cathode in an electrolytic cell supplied by an external electrical pressure. Various types of primary cell arrangements that could be employed in practice were classified. The efficiency of the cell for protection will depend on the current density at the cathode, and this will be controlled by the resistance of the cell and the effective voltage. The importance of overvoltage in determining the effective voltage was discussed.

The history of Sir Humphry Davy's application of zinc and iron protection for the prevention of the corrosion of the copper sheathing of ships and subsequent inventions for the protection of condensers and pipes were detailed. The patents of

[1]See this journal, November, 1912, p. 717.

Harris and Anderson, in which aluminium alloys are used for the prevention of the corrosion of condenser tubes, were shown to be primary cell methods. In the case of the use of zinc in boilers was involved a knowledge of the electrolytic resistance of the boiler waters and the effective voltage at temperatures from 150-200 deg. C., concerning which there is great need of experimental data. The amount of zinc used in some marine boilers is as great as from 400-600 lb. of rolled zinc per annum. If the zinc is efficient in producing electrical currents then the average current may be from 17 to 25 amperes. It was obvious that such currents would be obtained much more economically by the use of a dynamo.

The direct use of electrical currents has been the basis of a number of patents. Those of Mr. Elliott Cumberland were especially described. Iron anodes are placed in the water of the boiler, which latter is made the cathode. A low voltage of supply provided by a small motor-generator is used. The method has proved effective not only in preventing corrosion, but also in removing scale from the heating surface and preventing its formation. Experiments carried out at the Manchester School of Technology have shown that the current densities necessary to protect iron, copper, and other metals from corrosion of fresh and salt water are of low value, and hence in the cases of boilers and condensers the annual cost of electrical energy required is a small item. The chief cost will be in the renewal of anodes. Harris and Anderson have also applied electrical currents for the prevention of the corrosion of condensers. They find that a condenser with a cooling surface of 1025 sq. ft. requires only 2 volts and 2 amperes, and the special anodes used by them cost from £3 5s. per 1000 sq. ft. per annum. The use of electrical currents may also be applied in chemical works to prevent the corrosion of metallic screens and vessels from acid liquids.

Mr. **J. T. Crabtree** read a paper on "The Nature of Overvoltage." This may be regarded either as the excess of the anodic or cathodic decomposition voltage of a dilute acid with a given electrode over that for platinized-platinum, or as the excess of the back e.m.f. set up as an electrode (anode or cathode) after polarization over that set up by a platinized platinum plate under identical conditions.

This back e.m.f. was determined by alternately polarizing an electrode and measuring its single potential difference by means of a potentiometer, alternate connection between the latter and the primary circuit being made by means of a rotating commutator.

Overvoltage is affected by the time of application and current density of the polarizing current, the nature of the electrolyte and electrode employed, and by the thickness of the latter. Since these factors also affect the back e.m.f. of platinum, a comparison electrode was employed consisting of a thin deposit of platinum on Jena glass, the back e.m.f. of which is constant with time.

The overvoltage of a metal was taken as the difference between the back e.m.f. set up after an application of the polarizing current for thirty minutes at a given current density, in $\frac{N}{10}$ acid, and that set up by the above electrode under similar conditions.

It is probable that overvoltage is simply a manifestation of the difference between the rate of production of ions at the electrode and of the combination of these form complexes. Probably after being discharged the ions pass into a metastable condition and give rise to a back e.m.f. The catalytic activity of the electrode is probably an important factor as affecting this velocity of reaction, and the difference in catalytic activity of various metals under different conditions of texture, temperature, etc., would explain why the overvoltage varies largely with different metals under different conditions.

Overvoltage may be of importance in the case of the corrosion of a metal as either:

(*a*) Having a tendency to retard the deposition of hydrogen or oxygen on its surface, which might tend to assist or prevent corrosion.

(*b*) By setting up a high back e.m.f. which would diminish the effect of a decomposing current in cases of corrosion due to electrolysis.

(*c*) By assisting or preventing the solution of a metal. The condition that a metal may dissolve in an acid and evolve hydrogen is given by the relation

$$\text{e.m.f. of metal} - \text{back e.m.f. of hydrogen} > \eta$$

where η = overvoltage of metal. If η is large, no solution of the metal can occur. This is apparent with galvanized iron, where, owing to the overvoltage of the zinc, very little solution of iron takes place, the latter assuming the overvoltage of the external metal.

In view of the absence of any suitable factor to indicate the tendency of a metal to corrode, future experiments may indicate a parallelism between the overvoltage of a metal and its corrosion factor.

Dr. **W. Rosenhain** read a paper entitled "Note on a Specimen of Ancient Iron from Ceylon."

The reading of the paper was followed by a long discussion. A number of specimens of corroded metals was exhibited.

Electric Steel in Germany

In *Elektrotechnische Zeitschrift*, April 3, 1913, an interesting review of the present status of electrometallurgy in Germany is given by Victor Engelhardt, the chief engineer of the electrochemical department of the Siemens & Halske Company. As mentioned before in these columns, the large 25-ton Heroult furnaces have been put in operation on the Deutscher Kaiser. The largest size of induction furnace is at present 13 tons. As a result of the competition between induction furnaces and arc furnaces, the opinion seems now to have been firmly established that both types of furnaces have a right of existence. For this reason the Gesellschaft fur Elektrostahl Anlagen, which is affiliated with the Siemens & Halske interests and which owns the principal patents for induction furnaces, has now also acquired the rights for the Girod arc furnace for Germany. Otherwise, no change has come in the exploitation of the different furnace types by the large electrical manufacturing companies, as the Heroult-Lindenberg electric furnace interests are affiliated with the Allgem. Elek. Ges. and the Nathasius furnace interests with the Bergmann concern. The use of electric furnaces for melting ferromanganese has been widely introduced since the advantages are evident in this case. The cost of operation can be reduced so as to show a clear saving without taking into consideration the technical advantages which are doubtless in existence, but cannot be easily stated in exact figures. In the thirty large German steel works which use large quantities of ferromanganese in their converter plants, twelve had electric furnace plants for melting manganese in operation or course of erection at the end of 1912. Five of these plants were erected by the induction furnace-Girod interests, three use Heroult furnaces, two Keller furnaces, and two Nathasius furnaces. As to the use of the electric furnace for copper and zinc smelting, Mr. Engelhardt points out that the electric furnace has to meet in this field not only competition from the old-established metallurgical processes but also from leaching processes with electrolysis. While ten or fifteen years ago the potentialities of wet electrolytic processes were overrated, there is at present a tendency to pay new attention to their possibilities.

Gold and Silver

Cost of Dredging.—In the *Mining Magazine*, March, 1913, Mr. C. W. Purington gives, among other data on the Seward peninsula, Alaska, the cost of dredging at the Plein dredge, on Otter creek, tributary to the Nome river. At this dredge the ground is thawed by means of steam-points. The equipment for this purpose consists of two boilers, totaling 80 hp, mounted on skids covered with tents. They are moved ahead as fast as the dredge works the ground. Forty steam-points are used to thaw the ground to a depth of 14 ft. The duty of each point is 16 cu. yd. in 24 hours, and the cost is under 13 cents per yard. The cost of power generated, with oil costing 7.27 cents

per gallon delivered at the plant, is 3 cents per horsepower-hour.

The total cost of dredging and thawing at this plant is said to be 27 cents per cu. yd., handling about 800 yards per day. The cost at this plant, however, is not representative of what can be done on a larger scale. In the Nome tundra area, it is not impossible that, where a sufficiently large yardage is available and a proper equipment installed, the entire cost of stripping, steam-thawing and dredging will be under 20 cents per cu. yd.

Coarse Crushing in Cyanide Solution.—Some gold ores are readily amenable to cyanidation, even when crushed as coarse as ½ in., and when such is the case, it is possible to mill low-grade ore at a profit, even though the percentage extraction is not as high as that of some mills in which fine grinding is practiced. Two good examples of this class of ore, and of the improvement in milling made by changing from fine to coarse crushing, are given in the *Mexican Mining Journal*, March, 1913, by Mr. ERNEST H. SIMONDS.

The first ore is described as very porous, oxidized, and of medium milling grade. It would absorb and readily retain 15 per cent moisture. It was being crushed in stamps to 20-mesh and finer, the stamp duty being about 4 tons per day. About 55 per cent of the stamp product was coarse enough to be leached and the balance was agitated and filtered. Laboratory tests indicated that good extraction could be made with much coarser ore. One of the batteries was then fitted with a 4-mesh screen, and a lot of ore crushed. The stamp duty rose to 7.8 tons per 24 hours, and 76 per cent of the product could be leached. The extraction was as good as when ore was crushed to 20-mesh; the capacity of the mill was almost doubled by making an addition to the number of leaching tanks; and the slime department needed no enlarging.

The second ore was a medium hard quartz containing about 6 per cent of coarsely disseminated pyrite. Little slime was formed when the ore was crushed. Little concentrate could be obtained by crushing to 4-mesh, but at 8-mesh it was possible to secure a concentrate containing 73.5 per cent of the gold in the ore. At 12-mesh the concentrate yielded only 70 per cent of the gold in the ore. The final mill treatment was as follows: Crush to 8-mesh in cyanide solution, size the pulp on a 30 or 40-mesh screen and concentrate the oversize on jigs and the undersize on tables. All coarse and fine sand tailings are washed to recover cyanide solution, and then discharged to waste. The slime is thickened, agitated and filtered. Coarse crushing is expected to give a stamp duty of 7 tons if no regrinding apparatus is used, and more than 10 tons with regrinders. The test data show that the jigs will handle more than two-thirds of the ore; that the screen undersize, or sand size, will amount to less than one-quarter of the ore and contain more than one-half of the concentrate in it; that the same recovery can be made by coarse crushing and concentrating as by sliming all the ore to 150-mesh and cyaniding it without concentration.

The New Shamva Mill, Rhodesia.—The following outline of the equipment and process to be employed in the new Shamva mill in Rhodesia is interesting as showing the trend of milling in that country. It is taken from the *South African Mining Journal*. The ore is to be sized before going to the stamps, all ¼-in. particles being passed directly to the tube mills. The larger sizes will be crushed in cyanide solution by fifty-six 2000-lb. Nissen stamps fitted with ¼-in. mesh screens. Classifying cones will be placed ahead of the tube mills, and amalgamation plates and blankets after the tubes. The tube mill product is returned to the cones. The overflow of the tube mill cones will pass to eight sand cones superimposed above four others, where the sand will be classified and sent to leaching tanks. The slime overflowing the last cones will be thickened in 35 ft. x 10 ft. Dorr thickeners and agitated in Pachuca tanks, 45 ft. x 10 ft., connected in series. A Butters filter of 336 leaves will remove the valuable solution from the slime. Precipitation will be effected in ordinary zinc boxes, and the gold slime melted in gas-fired furnaces.

Copper

Historical Note on Gas-Fired Reverberatories.—The application of producer gas in firing reverberatory copper smelting furnaces is regarded as a recent development, although it is true that certain metallurgists foresaw some years ago the value of the method and predicted its extended use. It appears, further, from an article on mining in the Argentine Republic, by Mr. W. EVAN SIMPSON, in *Mining Magazine*, March, 1913, that one Fouert, "an unknown but progressive metallurgist," actually used gas-firing on a small reverberatory furnace at a place called Corrales, more than twenty years ago. "The plant consisted of a water-sealed producer, a reverberatory capable of holding a charge of about one ton, and a brick stove built on the best regenerative principles." A furnace charge required about six hours for reduction. The ore contained from 7 to 12 per cent copper, and the slags assayed only 0.5 per cent copper. The only fuel available for the gas producer was wild mountain bush, locally known as *jarilla*, which, though very resinous, is so small that difficulty is experienced in finding a stick of it larger than 1 in. in diameter. The apparent success of this venture in the wilds of South America, is worthy of the consideration in similar inaccessible localities.

Grinding-Pan Practice

Notes from the Great Fingall, Australia.—In the monthly *Journal* of the Chamber of Mines of West Australia, Mr. E. JENSEN, metallurgist for the Great Fingall company, presents some notes on the efficiency of the grinding pan, and of the relative merits of flat and corrugated shoes and dies. The latter first came to his notice about four years ago when he was metallurgist at the Oroya Black Range, through Mr. W. Jordan who was then mechanical engineer at the same mine. The first trial indicated a greater grinding efficiency, owing to the increased grinding surface offered by the corrugated surfaces. The corrugations are spaced 1½-in. centers, and take the form of concentric rings on the whole set of shoes and dies, each shoe and die being cast with a proportionate section of the concentric corrugations.

Mr. Jensen gives a number of tables showing the daily efficiency of the pans during the life of a set of shoes and dies. The life of a set of flat-surface shoes and dies appears to be about 21 days, while with corrugated surfaces the same sets lasted for 37 or 38 days. In calculating the grading analyses and grinding efficiencies in these tests, Mr. Jensen has used Stadler's method,[2] of which the following are the essential points. The mechanical work necessary for reducing the weight of a unit from one grade to the next following, is a constant for each grade, and is called the energy unit, denoted by E. U. The useful work done per unit by any crushing machine is the difference between the mechanical values of samples taken from the inlet and discharge of the machine. The relative mechanical efficiency is the value obtained by dividing the total work done by the horse-power consumed. The following table is a summary of the different tests made according to Stadler's formula:

	Test Numbers			
	1	2	3	4
Fan speed, r.p.m.	54	54	54	54
Shoe and die surface	flat	flat	corr.	corr.
Pan feed, E. U.	20.59	20.54	20.77	20.84
Pan discharge, E. U.	23.21	23.38	23.80	23.48
Work done, E. U.	2.62	2.84	3.03	2.64
Total E. U., 24 hours	99.82	107.92	118.78	111.41
Relative mechanical efficiency	10.51	11.36	12.49	11.73
Consumption iron per ton ore, lb.	3.75	3.56	3.04	2.88
Tons water per ton sand	5.33	5.33	5.50	5.10

"It appears that better results can be expected by having an increased grinding surface, as per medium of corrugated shoes and dies. Another factor worthy of consideration is pan speed. We are now undoubtedly getting, with a pan speed of 54 r.p.m., better results, with a lower power consumption per pan, than those obtained when the pans were running at 58 to 60 r.p.m."

[2]Grading Analyses and Their Application. Stadler. *Trans.* Inst. Min. & Met., Vol. XIX, pp. 471-485.

Recent Chemical and Metallurgical Patents

Gold and Silver

Treatment of Precious-Metal Ores Containing Copper.—Under certain conditions the presence of copper in gold and silver ores renders such ores not amenable to the cyanide process. According to the specifications of a patent recently granted to Mr. John Collins Clancy, of New York City, the use of a cyanamide in the cyanide solution overcomes the usual difficulties and makes it possible to treat successfully gold and silver ores by cyanidation. Mr. Collins has discovered that the addition of a cyanamide prevents the copper from consuming the cyanide, that is to say, the cyanamide forms an insoluble compound with the copper which is then soluble in excess of cyanide solution without destroying it. By precipitating the copper from the solution the cyanide can be regenerated and used as a solvent.

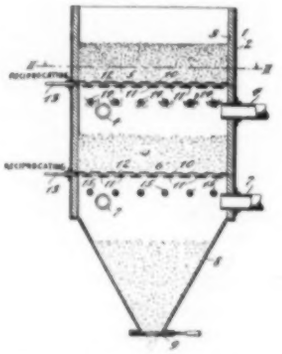


FIG. 1.—APPARATUS FOR TREATING ORES

Mr. Clancy's discovery further includes the fact that copper thus dissolved in a cyanide solution containing cyanamide, is not precipitated on metallic zinc, while the precious metals are thus deposited from the same solution.

The assumed reaction between copper cyanamide and cyanine is as follows: $nCuCN_2 + nKCN = (nCuCN,nKCN)$. The last complex compound is a solvent for precious metals and copper compounds. By simple electrolysis, it may be caused to yield regenerated cyanide.

As an example of carrying the invention into effect, the following figures are given by the inventor: 2000 lb. pulverized ore is suspended in 2000 lb. water containing 2 lb. sodium cyanide, 2 lb. calcium cyanamide and 2 lb. lime. This mixture is agitated for 8 or 12 hours, or until extraction is complete, after which the solution is displaced from the solid matter, and the precious metals therein contained are precipitated on zinc, without deposition of copper. The solution may be used over and over again until its copper content is such as to warrant removal of the copper by electrolysis. (1,057,936, April 1, 1913.)

Method of Treating Ores.—In Fig. 1 is shown a representation of an apparatus recently patented by Niels C. Christensen, Jr., of Salt Lake City, Utah, for the treatment of gold and silver ores. The apparatus comprises a novel form of roaster in which oxidizing, reducing, chlorodizing or sulphatizing processes can be carried on. The device consists of a shaft 1 having a suitable lining 3. Two floors or grates 5 and 6 are composed of staggered members 10 and 11, between which a reciprocating member 12, 13 can move for the purpose of delivering the ore continuously from one compartment to the next. The shaft is thus divided into compartments in which different conditions of temperature and chemical action can be maintained, and the ore successively subjected to these conditions. Parts 4 and 14 below the upper grate, and 7 and 15 below the lower grate, provide for the introduction of air, fuel, steam or other gas, to cause the proper reaction with the ore and added sodium chloride or sulphate or other salt.

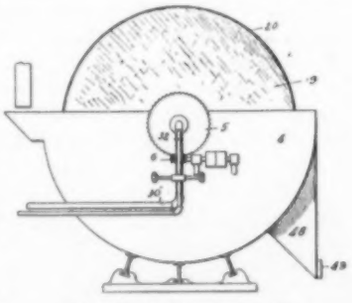


FIG. 2.—VACUUM FILTER

The inventor outlines various combinations that can be made in treating ore in this furnace, and gives results that he has obtained on ores from Colorado and Utah districts. "Practical tests have demonstrated the success and efficiency of this process of heating ores in stages at temperatures and for periods under positive control of the operator, and in atmospheres adapted to the particular kinds of ores treated." (1,058,034, April 8, 1913.)

Vacuum Filter.—A new form of continuous vacuum filter for ore slime is shown in Figs. 2 and 3, being the invention of Mr. George J. Young, of Reno, Nev. The device operates on the same general principles as other vacuum filters, viz., submerging a filter surface in slime pulp, withdrawing the valuable solution through the filter fabric, and building up a cake of slime on the fabric. These operations are performed successively and continuously on a plurality of disc-like filter units mounted on a hollow rotatable shaft, within which is a second hollow shaft which is stationary and which serves as a suction chamber, being connected with a vacuum pump. Suitable ports connect the different cells of a disc with source of reduced or excess pressure, so that vacuum and pressure can be applied alternately at the proper times.

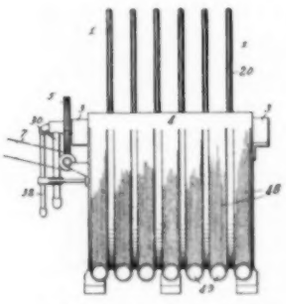


FIG. 3.—VACUUM FILTER

The apparatus consists of a slime tank 4, within which are mounted the filter discs 19. The latter are rotated by means of a gear drive as shown. The valves in the hub are so arranged that about seven-eighths of the disc is under reduced pressure, while one-eighth is subjected to excess pressure at the time of discharging the cake. The cake can be washed prior to discharge by means of sprays from pipes suitably arranged. Scrapers are used to divert the discharged cake into chutes 48, from which the slime is withdrawn through doors 49. (1,057,475, April 1, 1913.)

Agitator.—A modification of the Pachuca tank system of agitating ore slime has been patented by A. W. Warwick, of Colorado Springs, Colo. One form of his invention is shown in Fig. 4. It consists of a cylindrical tank a with a conical bottom a^1, in the center of which is supported an air-lift pipe B, supported at b, and capped at the top by b^1. Rows of holes b^2 are provided for the exit of the lifted slime. A source of compressed air is shown at c^1, connected with the nozzle C under the lift pipe, and with secondary air injectors d^2 arranged near the bottom of the conical bottom. The assumed level of material in the tank is the line X.

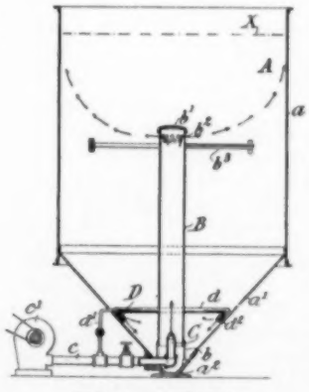


FIG. 4.—AGITATOR

In operation the compressed air lifts the slime through the central pipe, from which it issues through the holes in a lateral direction. The rising air serves to aerate the slime above the top of the lift-pipe, while the heavier solids fall to the bottom for further agitation. The auxiliary air injectors, near the bottom keep the slime there well aerated and agitated, and serve also to break up any cakes that may form during suspension of operations. (1,054,629, Feb. 25, 1913.)

Lead, Copper and Zinc

Electrolytic Production of Lead Hydrate.—According to a patent issued to Theodore G. Timby, of Chicago, Ill., lead ore may be fused and cast into bars or cakes of convenient form, and used as anodes in an electrolytic cell containing as electrolyte a neutral solution of sodium nitrate in water, producing under electrolysis lead hydrate. The lead ore is fused in a reducing atmosphere and cast into bars about 18 in. long,

12 in. wide and 2 in. thick. One or more such anodes are immersed in the electrolyte, with suitable cathodes, and electrolyzed. The following reactions show the cycle of operations.

$$NaNO_3 + H_2O = NaOH + HNO_3$$
$$2HNO_3 + PbS = S + Pb(NO_3)_2 + 2H$$
$$Pb(NO_3)_2 + 2NaOH = Pb(OH)_2 + 2NaNO_3$$

With a potential between one and six volts the current separated some free sulphur at the anode, forming a coating which can be brushed off, or which can be retained in the diaphragm when the latter is used. (1,056,382, March 18, 1913.)

Electrolysis of Argentiferous Lead Ores.—By ordinary smelting methods, silver-lead ore is reduced to silver-lead bullion which must be refined for the recovery of silver and the production of soft lead. It is proposed by THEODORE G. TIMBY, of Chicago, Ill., to fuse such ore and cast it into blocks that can be used as anodes in an electrolytic cell. The electrolyte is a hydrous solution of sodium nitrate, slightly acid with nitric or acetic acid to prevent the precipitation of lead hydrate. When a cell is thus equipped and electric current applied in the usual manner, silver and lead are liberated as a result of electrolysis. The lead may be collected upon the cathode in metallic form, or precipitated as, for example, lead chromate. The silver may be collected upon a filter cloth surrounding the anode, and into which is slowly fed a solution of alkaline halide, such as sodium chloride, which precipitates the silver. If no filter cloth is used, the precipitant may be fed into the bath between the electrodes and the deposited silver salt removed from the bottom of the cell. Some lead halide may temporarily precipitate, but will again dissolve and the lead will be deposited on the cathode. (1,056,383, March 18, 1913.)

Fusing and Purifying Copper.—A furnace and process for fusing cathode or electrolytic copper, and maintaining its purity during fusion, have been patented by MESSRS. WALTER S. ROCKEY and HILLIARY ELDRIDGE, of New York City. The object of the invention is to expose the metal to a reducing flame in the presence of the products of combustion, protecting the metal from oxidation after fusion by an atmosphere of the products of combustion, also while the metal is flowing from the fusion chamber to the ladle and from the ladle to the casting molds.

It has been found that electrolytic copper of high purity is contaminated by oxygen in the fusing process, forming oxides such as CuO and Cu_2O. By the process of the patentees these impurities are eliminated. (1,057,882-3, April 1, 1913.)

Electrolytic Recovery of Zinc and Manganese.—According to a process of ANDREW G. FRENCH, of Nelson, B. C., for the treatment of complex lead-zinc ores, the inventor obtains as a final result a strong aqueous solution of sulphate of zinc with neutral sulphate of sodium and more or less sulphate of manganese. This process is described in a former patent No. 1,041,060. More recent work by the same investigator has shown him that these three salts, viz., sulphates of zinc, sodium and manganese, are easily electrolyzed with the production of pure metallic zinc on the cathode and pure manganese dioxide on the anode.

The success of the process depends on the favorable combination of these three salts, and when insufficient manganese is present this element should be added.

The solution of the three sulphates is treated in an electrolytic tank having lead anodes and thin sheet zinc cathodes. Direct current is used, of from 2½ to 4½ volts with a current density of 20 to 30 amp. per square foot of cathode surface. Under these conditions pure zinc and manganese dioxide are formed. In the practice of the process, the solution flows continuously from the ore leaching tanks to the electrolytic tank, and after electrolysis and the removal of a portion of the metals, back to the leaching tanks.

The solution flowing into the electrolytic tank is practically neutral, while the electrolyzed solution is slightly acid and suitable for further reaction on ore. (1,055,157-8, March 4, 1913.)

Vanadium

Extraction of Vanadium from Carnotite.—SIEGFRIED FISCHER, of Golden, Colo., has recently patented a process for removing vanadium from carnotite concentrates or rich carnotite ores, leaving a residue containing all the uranium, radium and a negligible quantity of vanadium. The process consists in mixing the carnotite with caustic alkali in the ratio of about 5:1, adding water to form a plastic mass, and then evaporating and possibly re-evaporating until the reaction is complete, forming sodium vanadate. The mass is then lixiviated, the vanadium solution drawn off and made slightly acid with sulphuric acid, and the vanadium precipitated as iron vanadate by means of ferrous sulphate. This product, after drying can be reduced to ferrovanadium by the electric furnace or by the Goldschmidt process. Mr. Fischer's process was fully described by him in an article in this journal, August, 1912, page 469. (1,054,102, Feb. 25, 1913.)

Metal Alloys

Alloys for Cutting Tools.—In two patents granted to ELWOOD HAYNES, of Kokomo, Ind., the inventor describes new alloys which are valuable for forming cutting implements in which hardness, toughness and elasticity are demanded. The alloys also are capable of taking a high polish and receiving and retaining a sharp cutting edge. These alloys are formed from cobalt, chromium and one or more of the elements of the chromium group, such as tungsten, molybdenum and uranium. An example of a ternary alloy is one containing 15 per cent or less chromium, 15 per cent or less tungsten, and cobalt. Numerous other formulas are given in the patent specifications, for alloys having special application. (1,057,423-828, April 1, 1913.)

Government Specifications for the Purchase of Coal

Large users of coal will be interested in Bureau of Mines Bulletin No. 63, "Sampling of Coal Deliveries, and Types of Government Specifications for the Purchase of Coal," which has just been issued. The Federal Government, which purchases $8,000,000 worth of coal annually, buys more than half of it under specifications and has gone deeply into the question of sampling and analyzing coal. George S. Pope, the engineer in charge of such investigations and the author of the bulletin, makes the following statements:

To determine with utmost accuracy the ash content and heating value of a quantity of delivered coal would require the burning of the entire quantity, and special apparatus arranged to measure the total heat liberated, or would require crushing the whole quantity, and reducing it by an elaborate scheme of successive crushings, mixings, and fractional selections to portions weighing approximately 1 gram, the minute quantity which the chemist requires for each determination. Either of these procedures is obviously impracticable if the coal is to be used for the production of heat and power.

The method actually employed is to select portions from all parts of a consignment or delivery of coal and to systematically reduce the gross sample, obtained by mixing these portions, to quantities that the chemist requires for making ash determinations or that can be burned conveniently in the calorimeter, an apparatus for determining the heating value. The gross sample should be so large that the chance admixture of pieces of slate, bone coal, pyrite, or other impurities in an otherwise representative sample will affect but slightly the final results. Increasing the size of the gross sample tends toward accuracy, but the possible increase is limited by the cost of collection and reduction. In reducing the gross sample by successive crushings and halvings or fractional selections, the object is to procure a small laboratory sample that, upon analysis, will give approximately the same results as the gross sample itself, or, in fact, the entire quantity of coal from which the gross sample was obtained.

Recognizing the importance of the method of sampling as being a definite commercial procedure and of having the method clearly set forth in the specifications to become a part of the

contract, and recognizing also the desirability of insuring uniformity and similarity in the specifications used by the different branches of the federal service for the purchase of coal, representatives of the executive departments and independent establishments of the government held a conference under the auspices of the Bureau of Mines in February, 1912, for the purpose of discussing these and other features of the specifications. At this conference committees were appointed to prepare specifications in accordance with the views of the members. It was recognized at the conference that in general specifications, such as were recommended, certain requirements had to be of wide application, as the specifications cover such a wide variety of conditions, not only as to character and quality of coal but as to type of furnace equipment, size of deliveries, methods of delivering, etc.

The specifications which were used for the purchase of coal on the heat-unit basis prior to the fiscal year 1912-13 were on the B.t.u. (British thermal unit) "as received" basis; that is, payment for delivered coal was directly affected by the moisture content of the sample received by the laboratory. This method was based on the assumption that the moisture in the samples collected at the time of weighing and delivery could be preserved with slight loss during the storing and subsequent working down of the gross sample to a quantity convenient for transmittal to the laboratory and in its later treatment in the laboratory. From experiments that have been made and from a large mass of data, it is known that the moisture content of coal does not remain constant, and that the moisture content reported by the laboratory may be as much as a 5 to 10 per cent lower than that actually contained in excessively wet or high-moisture coal at the time of weighing.

* * * As a sample loses moisture, its B.t.u. "as received" value correspondingly rises, with the result that the price for delivered coal determined on the "as received" value is, with rare exceptions, higher than that warranted by the quality of the coal at the time of weighing. As a general statement, payment based on the "as received" B.t.u. value will be higher than warranted, unless the sampling and laboratory work can be carried on under conditions that minimize moisture loss, as under freezing temperatures.

Recognizing the uncertainty involved in taking the moisture determination in the laboratory as representative of the moisture content of the delivered coal and the consequent possibility of payment of a higher price than is warranted, the Bureau of Mines recommended to the executive departments and independent establishments of the federal service that the heating value in the coal specifications for the fiscal year 1912-13 be on the "dry coal" basis.

In preparing these specifications the fact was recognized that the amount of moisture contained in coal produced from day to day from the same mine, or group of mines working the same bed, is largely accidental, and is a matter over which the buyer and seller have only slight control. However, in order to place a negative value on high-moisture coals and to protect the government against the delivery of coals containing excessive amounts of moisture, the specifications require the bidders to specify the maximum moisture content in coal offered. This value becomes the standard of the contract.

If coal of uniform B.t.u. "dry coal" value is delivered on a contract, the contractor receives the advantage on any delivery in which the moisture content approaches the maximum specified, because he is paid for the weight of water contained in the coal in excess of a normal amount, whereas if the coal is very dry, containing less than the normal amount of moisture, the purchaser receives the advantage.

* * * As an example of the effect of a heavy rain on a car of coal in transit, a precipitation of 3 in. of water on a loaded 50-ton car, area of top about 360 sq. ft., would increase the weight of the coal 5.01 per cent, provided none of the water drained out or evaporated. It is obvious that if this coal is weighed and delivered immediately, special samples for moisture determinations should be collected and prepared at once and sent to the laboratory as a basis for equitable adjustment of payment on account of the excessive amount of water in the coal. As the weight of the coal was increased by the excess water, there should be a corresponding decrease in the price to be paid.

If a railroad car or wagon so rained on should not be unloaded immediately after weighing and special moisture samples should not be properly collected, prepared, and sent hermetically sealed to the laboratory, it is obvious that the purchaser would pay a higher price than warranted, especially if the car or wagon stood for some time before sampling and some of the water drained out. Further, if the coal was not immediately unloaded and sampled or if the car continued in transit after weighing, then the coal at the top would soon dry; and in either case the effect of the 3-in. rainfall, as indicated by the analysis, might be only a fractional percentage of the moisture contained in the coal at the time of weighing.

The determination of the moisture of coal delivered from stock piles is often of great importance, for the proportion of moisture contained in the small sizes, which are most abundant near the center of a stock pile and which absorb the rains, and melting snows in districts of heavy snows, may be from 10 to 15 per cent higher than when stocked. It is apparent, therefore, that special moisture sample determinations are necessary for equitable adjustment of payment on amount of excessive moisture in coal stocked in piles exposed to the weather.

The specifications provide for the collection of "special moisture samples" if, in the opinion of the government officials sampling it, the delivery contains moisture in excess of that guaranteed by the contractor. The "special moisture samples" are prepared in a manner to minimize moisture losses and may be taken and prepared independently of the gross samples collected for the determinations of heating value (B.t.u.), ash, and other specified data. If the analysis of the special sample shows a moisture content in excess of the contractor's guaranty, a proportionate deduction is made from the price to be paid for the coal.

The Joseph Dixon Crucible Company, of Jersey City, N. J., have issued a very complete "production catalog" of over 100 pages of type and illustrations, giving a full description of the entire Dixon graphite line. This catalog should be very effective in acquainting those who are already users of one form of graphite with its many other forms and uses.

Buffing and Grinding Motors.—The Charles J. Bogue Electric Company of New York City have placed on the market a new design of direct-current buffing and grinding motors in five sizes from ¾ to 5 hp, for connection to 110-volt and 220-volt circuits.

New Pulp Mill Project.—The Chesapeake Pulp & Paper Company, of West Point, Va., it is said, will have the first sulphate pulp mill in the United States built entirely by an American engineer with modern American methods of design and construction. This mill is now being built by Mr. George F. Hardy, New York; while Mr. Philip B. Sadtler, assistant general manager of the Swenson Evaporator Company, Chicago, has secured the contract for the entire machinery layout in the recovery end of the plant.

Zinc Dust Precipitation.—Experience with this form of precipitating cyanide solution at the Brakpan mill, South Africa, is related in *Bulletin* 101, Inst. Min & Met., by C. B. Brodigan. The equipment consisted of three Merrill presses with belt feeders. The latter caused trouble at first on account of irregular feeding, but this was overcome by alterations. Clear solution also was found essential, as slightly turbid solution choked the cloths, causing a rapid rise in pressure in the press, and necessitating by-passing and resting of the press. Strength of solution had to be higher than with zinc-box precipitation. There was a feeling of some anxiety about the reliability of precipitation, which led to the use of continuous testing. In spite of these troubles, which were largely incidental to starting a new process, it was generally believed that the press system offered advantages over the use of boxes.

Oil Extracting and Washing Centrifugal

It has so long been the custom of many plants and manufacturing establishments to regard the burning or discarding of their oily waste and other machinery-wiping material as a matter of course that they have ceased to consider what such a practice actually cost them, or, more properly speaking, perhaps have considered such expense as unavoidable.

The consumption of waste, towels and other wiping materials by every plant or manufacturing establishment is, of course, subject to the amount and character of its power and industrial equipment. However, when reckoned by the year it is by no means the least of the items of operating expense.

FIG. 1.—CENTRIFUGAL STEAM-TURBINE DRIVEN COMBINED OIL-EXTRACTING, WASHING AND DRYING MACHINE

When to this is added the value of the oil and grease that is lost in the discarded waste and other machinery-wiping material, the figure approaches a total that would surprise those who have not given the matter serious consideration.

This condition presents great opportunity for a successful process of waste and oil reclaiming, and where machinery-wiping towels are used a satisfactory method of cleaning them. Naturally such a promising field has not been neglected and several methods have been employed to reduce the loss of waste and oil, beginning with drip pans, presses, chemicals, etc., to more modern and efficient machines with increasing success.

The accompanying illustrations show the most recent and highly efficient and successful machine that has been devised for this purpose by the D'Olier Centrifugal Pump & Machine Company, of Philadelphia.

The machine is a combined oil-extracting and washing centrifugal, steam-turbine driven and manufactured in sizes 10-in., 15-in., 20-in. and 36-in. baskets.

The basket, which is entirely enclosed within the curb of the machine, is directly connected to the steam-turbine wheel. The following description of the operations of the machine can be readily understood by reference to the accompanying line drawing. The waste, towels or other machinery-wiping material to be reclaimed is placed in the basket and the steam turned on, rotating the basket at high speed. The small arrow heads indicate the course of the steam from the point of entrance (nozzle), through the turbine wheel into the bottom of the basket, thence through the waste, acting thereon, and out through the exhaust.

The effects of the steam on the waste, or other contents of the basket, are to heat and open up same, thinning the oil and grease, which in turn are thrown off through the perforations in the sides of the basket to the inside of the curb of the machine and are then conveyed by gravity through the oil trap to precipitate any solids and discharged into an oil receptacle. Heavy arrow heads indicate the course of the oil from the contents of the basket.

The waste or towels having been freed of oil and grease are then ready for cleaning; the water tap is opened and the basket and its contents submerged. Special washing powder is placed in the bath and by means of a steam jet the water is boiled while the basket is slowly revolved, thoroughly and continually agitating the contents. After the washing process, all free water is drained from the machine and the basket run at high speed to extract remaining moisture. The waste or towels are then ready for air-drying and further use. This reclaiming process can be repeated many times at a great saving of machinery-wiping materials.

The oil-reclaiming process will save from 90 to 95 per cent of the oil and grease in the soiled material. The complete process is thus one of dual economy in the recovery of oil and reclaiming of waste and towels. The economy thus effected can be readily appreciated.

This machine, by the use of an interchangeable basket, furnished by the manufacturers, can also be successfully used for the recovery of oil from automatic machined parts, metal chips and turnings, scrap metal, etc. Also, for general laundry purposes and like economic uses, and further, supplies a long-felt want in many laboratories for an efficient hand or power operated centrifugal for testing or experimental work.

The entire machine bears evidence of thorough and efficient design and construction. The manufacturers lay particular stress upon the enforced steam circulation whereby the steam acts on the entire contents of the basket; also upon the design and position of the main bearing, located above and not below the water level. This machine is also equipped for electric motor or belt drive for special service.

The D'Olier line includes a modern and highly efficient machine of the centrifugal type for the recovery of oil and grease from metal chips, turnings, scrap metal, etc., electric motor and belt driven. It can be fitted with an interchangeable basket for extracting oil from machinery-wiping materials for installations where steam is not available.

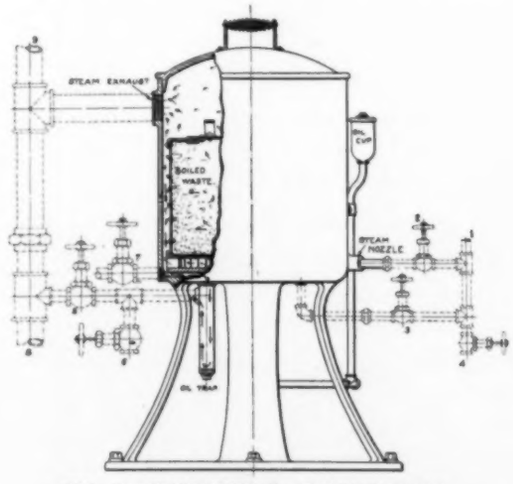


FIG. 2.—CROSS-SECTION OF CENTRIFUGAL

The machine is of the "Weston" self-balancing type, which is a decided advance in machine construction for this service.

The D'Olier Centrifugal Pump & Machine Company is continuing and enlarging its well-known line of centrifugal machines and pumps, manufactured for many years by D'Olier

Engineering Company, and is prepared to furnish centrifugal machines for every possible purpose, ranging from 10-in. to 72-in. baskets of varied design and for steam, water, electric and belt drive.

New Float-Type Recording Differential Pressure Gauge

In the April issue (page 222) a description was given of the spring pressure tube designs of recording differential pressure gauges made by The Bristol Company, of Waterbury, Conn. This company is now putting on the market another invention of its president and founder, Prof. Wm. H. Bristol, as illustrated in Fig. 1.

This new float type differential recorder was patented

FIG. 1.—FLOAT-TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

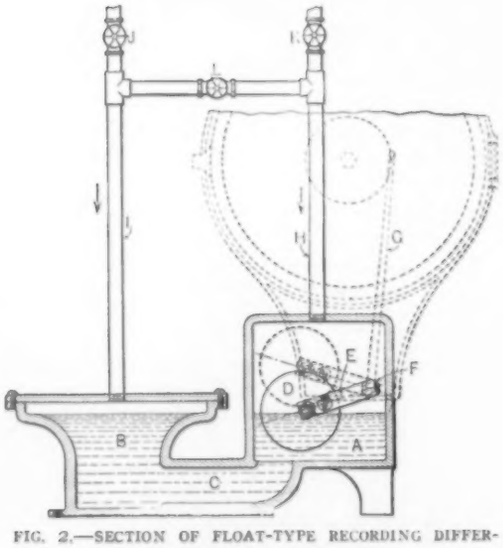


FIG. 2.—SECTION OF FLOAT-TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

August 15, 1911, and has been tested under practical operating conditions in steel works. It was developed to meet requirements for which the spring pressure types of differential recorders are not suitable, that is, for very low ranges of differential pressure in air, gases, liquids or steam, for applications where the static and differential pressures fluctuate rapidly, and also to satisfy the demand for instruments to record the rate of flow or volume directly on charts having uniform graduations in units of flow of volume.

The construction and principle of operation of the instrument will be readily understood by referring to the sectional diagram Fig. 2. There are two pressure chambers *A* and *B* intercommunicating through the connection *C*. A cylindrical float *D* is located in the pressure casing *A* as indicated, and is connected by arms to the shaft *F*, which extends through the casing. The recording arm *G* is directly connected to the end of the shaft *F*.

Connections are made by means of the pipes *H* and *I* between the pressure chambers *A* and *B* and the two pressures whose difference it is required to record. Mercury or water are employed in the pressure chambers according as the differential range to be recorded is high or low. When the higher pressure is applied to the chamber *B* through the pipe *I*, the level of the liquid in this chamber is lowered and that in the float chamber is raised, carrying the float and the attached recorder arm with it and making a record on a circular chart as it is revolved by a special clock at the desired speed.

By making cross-sections of the pressure chamber *B* of certain proportions as indicated, it is possible to produce a scale whose graduations are uniform for equal increments of flow or volume. The Bristol-Durand radii averaging instrument can therefore be used to obtain total flow or volume from the chart record of this instrument for a period of twenty-four hours.

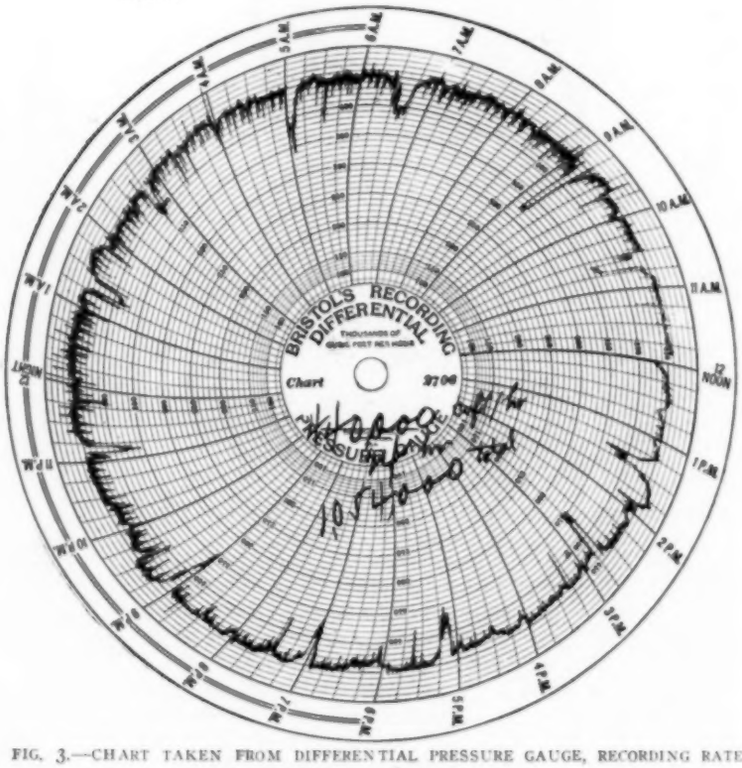


FIG. 3.—CHART TAKEN FROM DIFFERENTIAL PRESSURE GAUGE, RECORDING RATE OF FLOW AT BLAST FURNACE GAS

As these gauges will operate on extremely low differential ranges they are particularly well adapted for use with Pitot tubes for recording volumes of liquids, air or gases.

Fig. 3 is a reduced reproduction of a 24-hour record of the flow of blast furnace gas, made by one of these instruments in connection with Pitot tubes, at one of the largest blast furnaces in the United States. It will be noted that this chart is graduated in thousands of cubic feet of gas per hour, the range being from 0 to 500,000 cubic feet per hour. The working part of the chart above 150,000 is provided with uniform volume graduations. The actual differential pressure for the total range of the chart shown was 0 to 1.46 in. head of water. From this it will be seen that with a simple inexpensive Pitot tube it is possible to obtain sufficient differential pressures at slow rates of flow to directly record the amount and variations of the volume.

This instrument may be used to record the head of water in tanks under varying pressures, as, for instance, height of water in steam boilers and condensers. Another special application is that of recording the flow of water through a notch or over a weir, even though liquid is under pressure or a vacuum.

Magnetic Separation of Minerals

Although a number of machines have been designed and used successfully in the magnetic separation of minerals, less attention has been paid to the preparatory process of roasting. Recently this part of the work has been investigated more thoroughly, and a number of patents have been granted on apparatus and methods for making a proper magnetizing roast.

In the Campbell system of magnetic concentration the separation is accomplished by the conversion of the pyrite into a magnetic sulphide of iron in the Campbell furnace, and the subsequent removal of the magnetic sulphide by the Campbell separator. It is in the production of magnetic sulphide without loss of valuable metals in the roast that the Campbell process differs from all other magnetic separation processes. The operation of roasting to a magnetic sulphide is simple and easily controlled in the Campbell furnace. The pyrite is transformed superficially only, which means a minimum loss of sulphur, or gold, silver or copper which may reside in the pyrites. The resulting magnetic product obtained by the separator is therefore a valuable one for smelting or for the manufacture of sulphuric acid.

The Campbell roast is comparatively so slight that no metals are lost by volatilization, and a chalcopyrite ore, with gold or silver, may be roasted without material loss.

Both the furnace and the separator are small, light in weight, simple in construction, durable, automatic in operation and saving of power. Unlike other furnaces, this one can be started and stopped without affecting the character of the roasted product, and need not be operated continuously, but may be started in the morning and shut down at night regularly or may be run continuously.

The roasting begins five minutes after the oil fire is lighted in the roaster.

The roaster is without brick or other lining, and only the front section needs replacing after a considerable time, which replacement is made from duplicates kept on hand.

Chances of uneven roasting are largely eliminated. The heat is controlled by the amount of oil and air admitted, and the feed is controlled by an automatic feeder. The ore is presented to the heat always in the same way, consequently the ore is uniformly roasted and to just such a degree as may be desired.

Everything about the plant is automatic, and a shovel never touches the ore from the time it is unloaded from car or wagon. It is first passed through a revolving dryer, heated by fuel oil fire and elevated to a dry storage bin with hopper bottom. As required, it is discharged from the bin, elevated to a screen, the undersize of which passes to the roaster, oversize to rolls, and back to screen.

The roaster product is conveyed to a revolving cooler with jet of water inside, through cooler, and elevated to conveyor to feed separators.

The zinc from separators is conveyed by spiral conveyer and elevated to zinc storage bin. The pyrites from separators are conveyed in similar manner to iron bin. Both of these bins have hopper bottoms, and with conveyors and elevators discharge the ores into cars for shipment.

The process has been very successful in the treatment of the marcasite-blende concentrates in the Wisconsin field. A fully-equipped plant, capable of handling 75 tons per day, is situated at Cuba City, Wis. This plant is so arranged that lots of ore, varying from a few hundred pounds to carloads, can be treated successfully; also, a plant is now in operation at Linden, Wis., by the Linden Zinc Company.

The process is calculated to make a very high-grade zinc concentrate carrying a high percentage of the total zinc in the ore; also a pyritic product carrying a high percentage of sulphur and such gold, silver and copper as may reside in the pyrites.

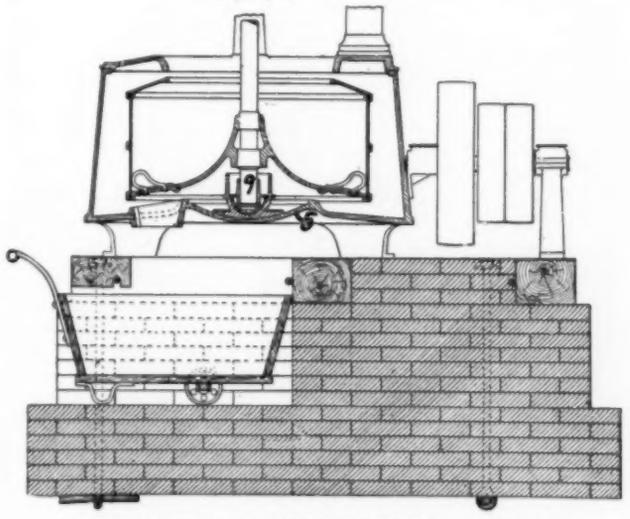
The Campbell Magnetic Separating Company, of Boston, Mass., in addition to the original Campbell patents covering the processes and apparatus for concentrating ores, has recently acquired other valuable patents relating to the magnetization of iron sulphides and apparatus for accomplishing the same.

A Self-Balancing Centrifugal

There has been much written on the subject of centrifugals, especially on their applications in certain chemical industries, but there still remains an important factor to be considered. That is, the question of taking care of an unbalanced load. As all users of centrifugals know, a basket or retainer out of balance to any extent when running at normal speed will rack itself to pieces in a short time or will do some damage to the building in which it is in operation.

The adjoining illustration shows a section of one of the various types of centrifugals built by the Tolhurst Machine Works, of Troy, N. Y., all of which are designed to be automatically self-balancing. For this purpose the Tolhurst centrifugal does not depend upon bumpers, rubber rings or springs, but by a very ingenious construction its basket and load supplies the means of balancing through the force of gravity in a flexible bearing that absorbs every particle of vibration.

The step in which the shaft or spindle 9 revolves, resting in the saucer of base 5, is free to adjust itself to any position taken by the rotating spindle, but as the saucer forms a quick circle in relation to the top bearing, the step when leaving the center meets the resistance or force of gravity which is always tending to pull the step back to the lowest point of the saucer.



SECTIONAL VIEW OF BOTTOM DISCHARGE EXTRACTOR

In this way all vibration is absorbed, the movable or flexible step taking all the shock.

The heavier the load the smoother it will run, although it may be 5 to 15 lb. out of balance.

The illustration shows the Tolhurst self-balancing hydro-extractor in the special type which is arranged with bottom discharge. For such subjects as sugar, salts, crystals, etc., this method of unloading is very desirable. The foundation may be so constructed as to allow a truck to stand under the extractor.

After the liquor has been removed from the materials, the lid is raised, and the material falls down into a receptacle, thus doing away with the necessity of shovelling it out.

The Tolhurst Machine Works build centrifugals with the self-balancing feature for all kinds of work, especially for use in chemical laboratories, chemical works, laundries, silk mills, etc., and suitable for any kind of drive, also under-driven centrifugals having removable baskets specially designed for nitrated cotton or chemicals that must be kept free from any and all contaminations.

According to the chemical nature of the material to be treated the basket and case may be of plain metal of nearly any required composition.

The basket may be rubber-covered and the case may be lined with porcelain, lead or rubber.

A New Foundry Laboratory

The new laboratories of The H. M. Lane Company were formally opened by a reception to the Detroit foundrymen on March 22, which was attended by about one hundred people.

These laboratories are the outcome of the research work which has been carried on by Mr. H. M. Lane during the past dozen years.

They are located at 18 East Piquette Ave., Detroit, Mich., and occupy some 5000 square feet of floor space.

Mr. E. J. Woodison, of The E. J. Woodison Company, cooperated with the laboratory staff, furnishing the foundry supplies necessary in the laboratory and the material for the lunch.

Refreshments were served from six to seven o'clock, the menu consisting of baked beans, baked potatoes, baked sausages, rolls, pickles, doughnuts, and coffee. Everything that could be baked was baked in a core oven, and the hosts even declared that the doughnuts were cooked in core oil. The oven used was a Crawford gas-fired oven, and the temperature was regulated by a number of attached thermometers furnished by the Taylor Instrument Companies, of Rochester, N. Y. Each man's lunch was placed in a new E. J. Woodison 16-qt. foundry pail, and, according to the most modern core-room methods, these were delivered to the visiting foundrymen on this occasion by

A CORE MAKER'S LUNCH

a gravity carrier. This full dinner bucket scheme goes the serve-self lunch one better, as the bucket first delivers the lunch and then becomes the diner's seat.

As to the plant and equipment, the following notes will be of interest.

At the front of the building there are located the offices and draughting room, back of these there is the chemical laboratory, which is 15 by 40 feet. The remainder of the main building and the adjoining building are given up to the working core room and model foundry plant, containing the latest appliances in foundry equipment.

The objects of the laboratory are: To study core sands and core binders, so as to find the best mixture for the conditions present in different plants; to study new core binders in order to ascertain their exact value to the foundry; to determine the best method of mixing sands and binders for different classes of cores or molds. A careful study is also made of the proper baking temperatures, baking times, and different core oven fuels.

Special machinery is being installed in the laboratory for cleaning and recovering old core sand.

In the foundry department there is a melting furnace and appliances for pouring molds so as to test both the cores and the molds.

A room at the rear of the building is fitted with microscopes and cameras for the study of core problems.

A permanent exhibit of apparatus and supplies is also provided and for this purpose Mr. H. M. Lane has had the cooperation of a large number of manufacturers.

A Proposed Standard Method for Sampling Ferro-Vanadium

By Wm. W. Clark

It has been requested that a standard method for sampling ferro-vanadium be given, as considerable discrepancies in the analysis result from poor and careless sampling. It seems that the average chemist goes on the supposition that if one or two pieces are taken from each keg of alloy (ferro-vanadium comes on the market in kegs weighing from 150 to 200 lbs. each), that this sample is a good representative average of the material. Such is not the case.

Segregation is a commonly known term, and has caused considerable discussion in the steel business, where a few tenths or even hundredths of a per cent of one constituent are present and segregate considerably. How much more will segregation be when elements are present up to 40 per cent, such as vanadium, or to 2 or 3 per cent, such as aluminum or silicon?

Commercial ferro-vanadium, like every other complex alloy, on cooling, goes through a solidifying interval and not a sharp solidifying point. The aggregate solidifying first will rise or sink in the molten mother liquor, according to the difference between its specific gravity and that of the molten magma. The next and successive solidifying aggregates will follow the same course until the remaining molten mother liquor will have reached the eutectic composition, when it will solidify as a whole. During this process of solidification, it is thus apparent that the chances for the separation of the various constituents are very great, especially if the solidifying interval stretches over a great range of temperature, and if the difference in specific gravity of the crystallizing constituents is great.

Besides this tendency of the alloy to separate various constituents on cooling, we have to bear in mind the fact that ferro-vanadium is manufactured in small quantities in crucibles, which will naturally vary in composition from crucible to a certain extent, and will thus enhance the importance of careful sampling for obtaining the true average of a large quantity.

The method used in sampling is one which has been adopted after numerous experiments, and has been found to give uniform results on checking. The alloy is shipped in several different sizes, going from lumps 6 inches in diameter down to 10 mesh, according to requirements. When sampling the lumps a couple of chips are knocked off each lump with a chipping hammer. The regular size, which is crushed to lumps about the size of a walnut, is sampled during the process of kegging by running the hand along the front edge of each shovelful of the alloy, which has been well mixed after crushing, and taking off all that can be held in one hand of both fine and coarse material. The finer grades of alloy are sampled by mixing and quartering until a sample is obtained of the requisite size. A sample of at least 3 per cent of the bulk of the alloy should always be taken; however, when the alloy is very coarse this should be increased considerably.

The samples are first put down to rice size in a small crusher, the jaws of which are hardened steel. This crushed sample is quartered twice, and then put down to 20 mesh. This is quartered until about 2 ounces remain, which is all powdered in a hardened steel mortar, until it will pass through a 100-mesh sieve. This sample is well mixed for use in the laboratory.

It might be well to mention here the fact that ferro-vanadium is not uniformly magnetic, so that if a magnetic spatula is used in weighing weird results are the rule.

American Vanadium Co.,
Bridgeville, Pa.

Mexico has always been the world's largest producer of silver, the next largest output coming from the United States, Canada, Australia and South America.

A New Molten-Metal Pyrometer

The Brown Instrument Company, of Philadelphia, manufacturers of pyrometers, are placing on the market an improved type of thermo-electric pyrometer for measuring the tempera-

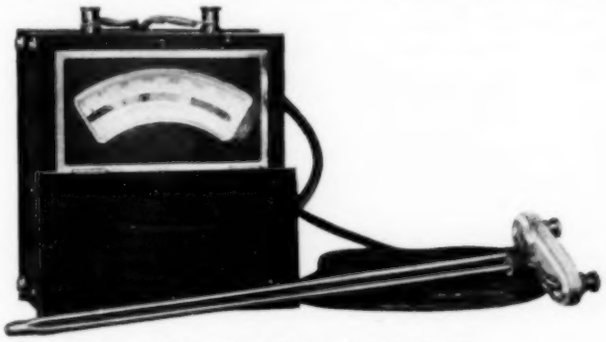


FIG. 1.—MOLTEN-METAL PYROMETER

ture of metals such as copper, brass aluminum, bronze, etc., in their molten state.

This company has been experimenting for some time in an effort to produce a thermo-couple which will withstand for a reasonable length of time the action of the molten metal. They have been able to produce a thermo-couple formed of nickel

FIG. 2.—THERMO-COUPLE

alloy rods ¼ in. in diameter which are inserted directly into the metal. On account of this large cross section (Fig. 1) in comparison with thermo-couples heretofore used which have a cross section of about ⅛ in. in diameter only, the life of the thermo-couple has been greatly increased.

In conjunction with this thermo-couple a high-resistance

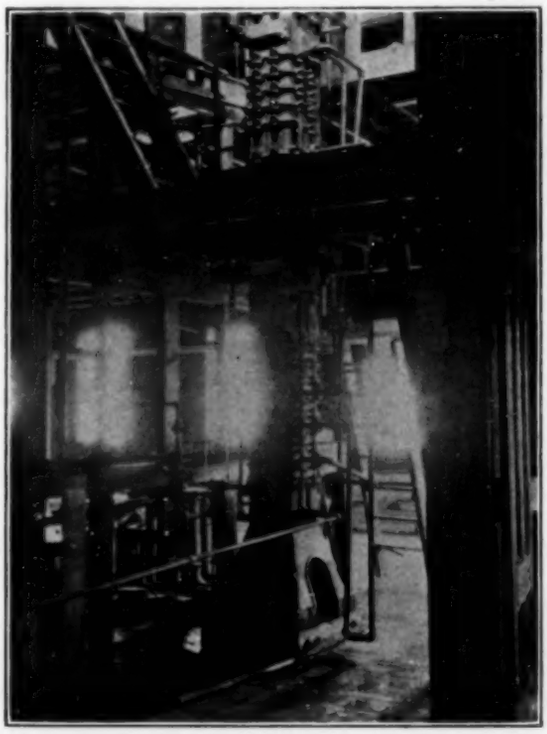


FIG. 1.—AUTOMATIC CONTINUOUS DISTILLING MACHINE

portable indicating instrument is used which is unaffected by changes in the length of the wire connecting the thermo-couple and the instrument, and which is sufficiently robust in construction for ordinary foundry service.

It has long been recognized that if a reasonable life could be secured from a pyrometer, all foundries would adopt them on account of the importance of knowing the temperature at which the metal should be poured. In the case of this new Brown pyrometer the thermo-couple is inserted directly into the molten metal before pouring, and the temperature instantly noted. When it has once been definitely determined what the proper pouring temperature is for a certain metal a concern can reproduce results continually, producing uniformally good castings from maintaining the same temperature.

Distilling Machinery

In our April issue, page 224, we noted the new industrial laboratory and demonstration plant of Mr. Walter E. Lummus, of Boston, Mass. The accompanying illustrations show some interesting machines installed in that laboratory.

Figs. 1 and 2 show an automatic continuous distilling machine for refining turpentine produced by destructive distilla-

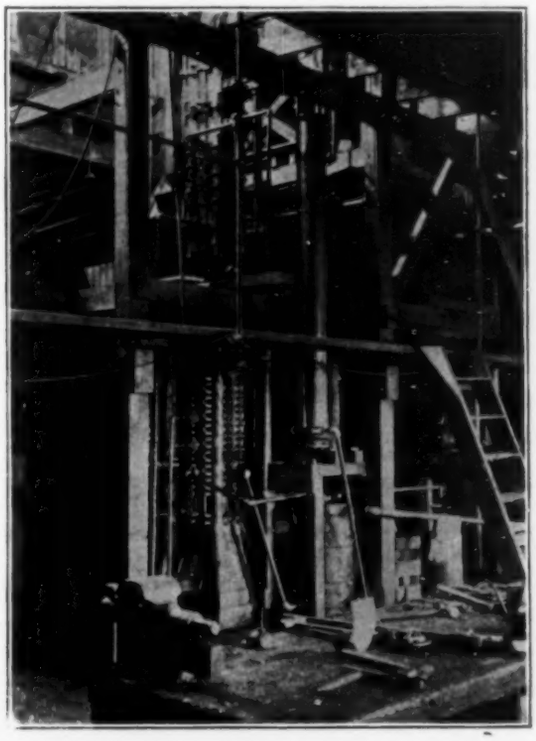


FIG. 2.—ERECTING SHOP, ASSEMBLING AN AUTOMATIC CONTINUOUS DISTILLING MACHINE

tion of long-leaf pine. Turpentine and pine oil are completely refined in a single operation. Fig. 2 shows the assembling of this machine in the erecting shop.

The continuous distilling machine shown in the cuts illustrates a small refining unit, equipped with positive controlling devices that regulate temperature, pressure, and cooling water, with absolute precision, obtainable for brief intervals in apparatus that is regulated by hand.

The machines are generally equipped with heat-exchanging devices or recuperators that utilize waste heat to warm the incoming raw material.

These distilling machines combine simplicity with large yields of products of uniform quality.

The economy of steam, in the case of very dilute or low-grade material, is sometimes equal to 80 per cent of the total that would be required with ordinary stills. The capacities range from 5 to 500 gallons per hour.

Figs. 3 and 4 show a continuous extracting and distilling outfit for simultaneous extraction of gums, oils, etc., and the recovery of the solvent in dehydrated or rectified condition. Fig. 4 shows the dumping, the extractor being inverted to discharge the spent material.

This experimental apparatus is capable of handling a great variety of materials by a diversity of processes.

It consists of a copper extraction cylinder of the dumping type, capacity 1 cubic foot, equipped with juice warmer and connected to tanks for supplying the solvent, and collecting the extract by gravity, and may be operated by either upward

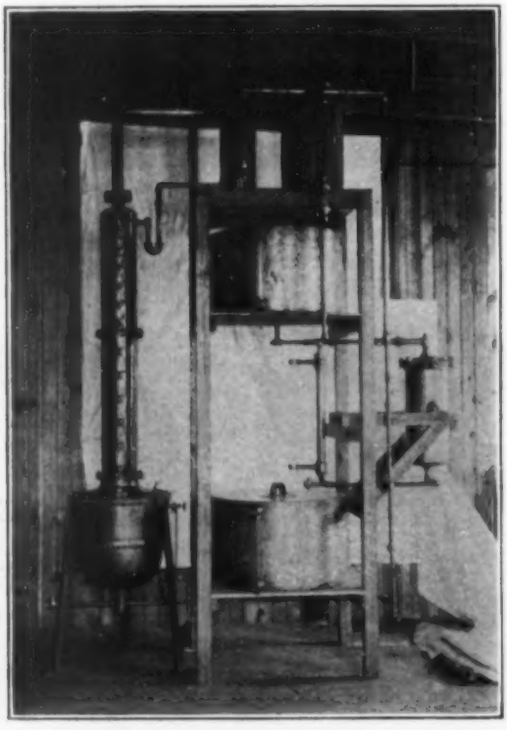


FIG. 3.—CONTINUOUS EXTRACTING AND DISTILLING OUTFIT

or downward extraction, in batches or by continuous process, with precise temperature regulation.

A still serves to concentrate the extract and recover the solvent. The still is provided with a plain still head and also with a very efficient rectifying column, the latter serving to repurify solvents that have become dilute or impure.

The extractor and still may be operated independently or in circuit by manipulation of valves.

The Lummus laboratory has 4000 sq. ft. floor space. There are two rooms for the assembling of distilling apparatus, 22 ft. high and 40 ft. long, also laboratories and work rooms. In the center of the building there is a trestle for supporting tanks and storing materials. The tanks are at different levels so that liquids may be run by gravity.

Occupational Diseases in Chemical Trades

A committee on occupational diseases in the chemical trades was appointed by the New York Section of the American Chemical Society in February, 1912.

The objects of the committee may be specifically stated as follows:

1. To hold itself ready to advise the Legislature of the States of New York and New Jersey in reference to matters pertaining to occupational diseases in the chemical trades.

2. To study various bills presented in the Legislatures in an effort to avoid unwise legislation; especially that which might be inoperative or ineffective from one or many reasons resulting from lack of technical knowledge at the time of writing the laws.

3. To inaugurate and superintend such investigations as might be decided upon which look toward improvement of conditions of labor in the chemical trades.

The personnel of the committee may be seen by a perusal of the membership, which is given herewith.

Dr. Charles Baskerville, Professor of Chemistry and Director of the Laboratory, College City of New York, chairman.

Mr. R. C. Uhlig, Chief Chemist, Brooklyn Union Gas Company, Brooklyn, N. Y., secretary.

Dr. Geo. P. Adamson, Baker & Adamson Chemical Company, Easton, Pa.

Mr. W. H. Bassett, Technical Superintndent and Metallurgist, American Brass Company, Waterbury, Conn.

Dr. Wm. F. Doerflinger, Consulting Chemist, 52 Beaver Street, New York City.

Dr. H. M. Kaufman, General Manager Mutual Chemical Company of America, 55 John Street, New York.

Dr. Chas. F. McKenna, Chemical Engineer, 50 Church Street, New York.

Dr. A. C. Langmuir, Chief Chemist, Marx & Rawolle, 9 Van Brunt Street, Brooklyn, N. Y.

Dr. Charles L. Parsons, Mineral Chemist, Bureau of Mines, Washington, D. C.

Dr. Geo. A. Prochazka, General Manager, Central Dye Stuff & Chemical Company, Newark, N. J.

Dr. Geo. D. Rosengarten, Powers, Weightman & Rosengarten, Philadelphia, Pa.

Mr. A. H. Sabin, Consulting Chemist, National Lead Company, 129 York Street, Brooklyn, N. Y.

To date the following work has been undertaken or completed, namely:

Dr. Charles L. Parsons, Mineral Chemist of the Bureau of Mines, Washington, D. C., reports that the Bureau's experts engaged in studying mineral resources and utilization under instructions from Dr. J. A. Holmes, Director of the Bureau of Mines, are noting health conditions and collecting information on occupational diseases in chemical trades in so far as they relate to mining and metallurgy. When funds are available, bulletins will be issued on the subject from time to time.

Dr. Charles F. McKenna was retained by the Factory Investigating Commission of the State of New York, as chemical

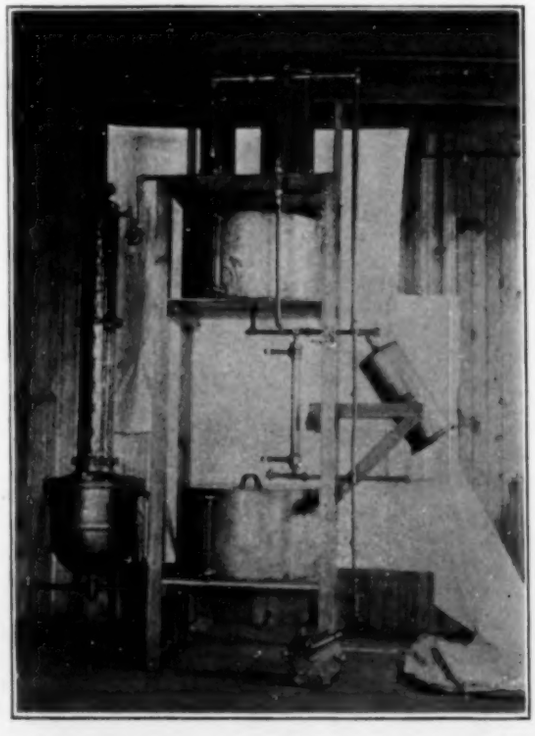


FIG. 4.—EXTRACTOR INVERTED TO DISCHARGE SPENT MATERIAL

adviser to Dr. Charles M. Price, Medical Director, who supervised the investigation of the chemical trades in New York State. Dr. McKenna's work was associated with the manufacture of "commercial acids."

Dr. Charles Baskerville prepared a report for the Factory Investigating Commission of New York on "Wood Alcohol."

These reports, which are now in press and will soon be available, contained suggestions for some new legislation. Bills have been introduced carrying these suggestions. Among them is one which involves the appointment of a chemical engineer as one of the four members of the Division of Industrial Hygiene of the proposed reorganized Department of Labor.

Vulcanized Fibre for Chemical and Electrical Purposes

Vulcanized fibre—a modified form of pure cotton cellulose—was first made in this country some forty years ago in Pennsylvania and West Virginia. In 1873 a plant was erected in Wilmington, Delaware, and since that time the American Vulcanized Fibre Company, of Wilmington, has developed the uses of this material for the most various purposes, for fibre trunks, fibre suit cases, fibre washers, for spigot packing and fibre valves, for telephone insulation, for gears, frictions, bearings, etc.

These different applications are made possible by the characteristic properties of vulcanized fibre which has the consistency of horn, has great mechanical strength, is tough, pliable, can be turned, bent, embossed, drilled, threaded, easily machined, and is insoluble in all ordinary solvents.

Its low electrical and thermal conductivity makes it the best insulating material.

With respect to chemical influences vulcanized fibre is said to be insoluble in all ordinary solvents, and is not deleteriously affected by alcohol, turpentine, naptha, benzine, petroleum, or any of the animal, vegetable, or mineral oils, and by very few ordinary chemicals.

The special shapes in which it is manufactured include gaskets for steam, water, and oil; washers, gears, gear blanks, switch bars, and bases, valves, packings, etc., also tubes, sheets, and rods.

Personal

Dr. **Edward G. Acheson,** on his recent visit to Russia for the purpose of lecturing before the Russian Imperial Technological Institute on the nature and uses of deflocculated graphite and oildag and aquadag, was the guest of honor at a dinner at the American Embassy in St. Petersburgh. The Russian government made the dinner a special demonstration of friendship between the United States and Russia. Count Witte and Premier Kokovtsoff, who seldom meet, were both present. Premier Kokovtsoff, with the Czar's approval, treated Dr. Acheson as a State visitor. Dr. Acheson was presented with signed photographs of several members of the nobility, while the Czar had conferred to him the insignia and sash of the Order of St. Ann. This order compares with the highest orders conferred in other European countries and conveys nobility to the recipient in the Russian empire.

Mr. Kosaku Asano, metallurgical engineer for the Ashio copper mines, Japan, has returned to that country after making a tour of the principal mines and metallurgical plants in the United States.

Mr. Frank C. Bowman, metallurgist for the New Reliance Company, Trojan, S. D., has returned to the Black Hills from a visit to Denver.

Mr. William A. Carlyle, until lately professor of metallurgy at the Royal School of Mines, London, has resigned his position and formed a commercial partnership with Mr. John F. Allan. Consulting engineering offices have been opened at 62 London Wall, E. C., London, England.

The firm of **Emory & Eisenbrey,** civil, chemical and industrial engineers, 2 South Fifteenth Street, Philadelphia, Pa., announce that Mr. J. R. Taft, formerly with Coverdale & Company, Inc., New York City, has become a member of the firm. A New York office of the firm will be conducted at 50 Church Street.

Mr. **Milton M. Kohn,** the inventor of the multiple replaceable unit system of electric laboratory furnace, has just returned from a business trip to Europe.

E. J. Lavino & Co., of Philadelphia, Pa., importers of ores and ferro-alloys have been appointed sole American agents for the Société Anonyme des Bauxites de France, who are the largest producers of bauxite in France, and although they have been without previous representation in the United States it is said that they have been shipping from France the bulk of the bauxite which has come to the United States.

Mr. Henry Schnitzel, general manager for the Golden Reward Mining Company, Deadwood, S. D., was in Denver recently, enroute to New York City on a business trip.

Mr. Edwin G. Steele, of the engineering firm of Sutton, Steele & Steele, Dallas, Tex., has returned to Dallas after spending several weeks in Leadville and Denver, Col., investigating the ore situation at the former place, and supervising the erection of a large testing plant at the latter.

Mr. Charles E. VanBarneveld, until recently professor of mining in the University of Minnesota, has been appointed chief of the Department of Mines and Metallurgy for the Panama-Pacific International Exposition, at San Francisco, 1915. Mr. Van Barneveld is a graduate of McGill University and well qualified for the position.

Mr. Leslie H. Webb, secretary of the Wedge Mechanical Furnace Company, of Philadelphia, has returned home from a two-months trip to the Pacific Coast.

Mr. Pope Yeatman, consulting engineer for the Braden Copper Company, Chile, South America, is at the property on a trip of inspection. He will also visit the property of the Chile Exploration Company at Chuquicamata.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

703,857, July 1, 1902, Constantin Jean Tossizza, of Paris, France.

Refers to his application for patent 710,346, and relates to the electrolytic deposition of such metals as zinc, nickel, cobalt, and cadmium, from their sulphates, by charging the solution with sulphurous acid and electrolyzing under conditions that do not decompose this acid or liberate hydrogen. With zinc sulphate soluble anodes of pure copper are used, metallic zinc separating at the cathode, while copper sulphate forms at the anode; the voltage necessary for the operation is 1 1/10, and with currents of 40 amperes per square meter rises to about 1 5/10 volts. The copper sulphate is subsequently saturated with sulphurous acid and electrolyzed with 31/100 volt. The liquid remaining from the copper electrolysis serves to dissolve more zinc ore.

704,639, July 15, 1902, Carl Hoepfner, of Frankfort-on-the-Main, Germany; Henry Orth, Jr., administrator of said Hoepfner, deceased.

Refers to his prior patent 507,130, and relates to the extraction of metals, such as copper, from their ores, by a warm solution of cupric chloride containing less than its saturation quantity of alkaline chloride. This solution will dissolve the lead and silver from the ore, which are precipitated by refrigeration, and the cuprous chloride formed by the solution of the lead and silver is converted by chlorine into cupric chloride. The iron present is precipitated by oxy-chloride of copper, and the remaining solution heated and returned to the ores to dissolve the copper with some zinc. Any lead present may be precipitated by zinc oxide. The copper and zinc are now preferably precipitated electrolytically.

706,436, August 5, 1902, Frank Townsend Mumford, of Kalgoorlie, Western Australia, Australia.

Relates to apparatus for the treatment of electrolysis of ores and slimes, also clay-like and slimy ores, or such as require

fine crushing, of the precious metals. The apparatus consists of a long steel cylinder lined with copper which is amalgamated, and has a layer of mercury at the bottom. Through the center of the cylinder and concentric therewith are a number of carbon or iron rods connected as anodes; the copper lining serving as the cathode. The electrolyte may be potassium or sodium cyanide, chloride, or bromide, or a mixture of them. It should preferably contain a salt capable of yielding oxygen to accelerate the solution of the gold. The pulp is discharged after the operation is completed, the mercury removed and the gold amalgam treated in the usual manner to recover the gold.

709,817, September 23, 1902, Clinton Emerson Dolbear, of Boston, Massachusetts; assignor to American Mining and Metal Extraction Company, of Boston, Massachusetts.

Relates to a method of electrolytically extracting metal from ores such as copper, etc. The ore is suitably crushed and electrolyzed in a vat with an electrolyte consisting of sulphuric acid containing nitric acid, or a nitrate, such as sodium nitrate. Insoluble anodes, preferably of carbon are used, and a cathode of the same metal to be extracted. During the process the nitric acid dissolves the ore, the oxygen liberated by electrolysis oxidizing the lower oxides of nitrogen back to nitric acid, whereby no nitric acid is lost; the metallic nitrate formed is converted into sulphate, from which a reguline deposit of metal can be obtained on the cathode.

710,346, September 30, 1902, Constantin Jean Tossizza, of Paris, France.

Relates to the electrolytic deposition of copper or other metals from solutions of their sulphates using insoluble anodes and depolarizing the anode by sulphurous acid, or the introduction of sulphur dioxide gas through a hollow anode. The roasted ore is dissolved in sulphuric acid, and the sulphur dioxide given off in the roasting is collected and used for depolarizing the anode. The solution of ore obtained is electrolyzed with a voltage of 6/10 of a volt, whereas heretofore electrolysis of such a solution with insoluble anodes required at least 1 3/10 volts.

714,598, November 25, 1902, Sidney Theodore Muffly, of Bowdre, Georgia; assignor of one-half to Runyon Pyatt, of New York, N. Y.

Relates to an apparatus for the separation of metals from their solutions by electrodeposition, and refers to the application for his process patent 714,599. The apparatus consists of a plurality of precipitation boxes, in cascade, with the electrolyte, flowing through all the boxes in series, a filled box being removed from the beginning, and a fresh, empty one inserted at the end of the series. The boxes are divided into two compartments, one constituting a conduit, and the other an electrolytic compartment. The latter contains an anode of iron or other suitable material, and a mattress-cathode made of cellular porous carbon casings containing a carbon plate, and filled with a filiform packing of lead and zinc composition. The electrolyte consisting of a cyanide solution containing gold and silver, or a similar solution, flows through the box, down the conduit, and up through the mattress-cathode, in which the metals are deposited both by the electrolysis and also by the filiform lead-zinc packing. Compressed air is blown through the cathode during deposition. From this cathode, the solution overflows into another similar box.

Book Reviews

The Metallurgic Industry in Italy.—Description of the principal works. LXI and 430 pages, 7 x 10. Associatione fra gli Industriali Metallurgici Italiani, Milano.

This book was written in honor of the (British) Iron and Steel Institute and was intended for distribution among those of its members who would join the trip to Italy, in connection with the industrial exposition at Turin. It will be remembered that after all plans and preparations had been made, this trip of the Institute to Italy was finally abandoned.

The young and flourishing Italian industries had done everything in their power to tender a worthy reception to their British guests, and the above "Associatione" or, as we might say, the "Italian Society of Industrial Metallurgists," had prepared this instructive description of the country and the industry, with details on nearly every plant, where the metallurgical profession helps producing goods for commercial purposes or for national defense. The book is beautifully illustrated.

Although it is written in English we cannot help mentioning that we would mark this book with the mysterious sign of the cable companies L C O ("language of the country of origin") and not L C D as it is intended. The reviewer being acquainted with the Italian language had no trouble in understanding the text, and after all, what counts in a book of this kind, is the material—and this is all in the book—and the fine spirit in which it is written.

An Expensive Experiment: The Hydro-Electric Power Commission of Ontario. By **Reginald Pelham Bolton.** New York: The Baker & Taylor Co., $1.25.

This book is an amplification of the author's evidence before the New York State Committee on the subject of the activities, operation and results of the Hydro-Electric Power Commission of Ontario. Its appearance is particularly timely on account of the bearing of the subject on the schemes at present being advanced for the entrance of the State of New York into the business of power development and supply.

While giving the Ontario Commission credit for conducting "a difficult problem in an honest and sincere manner," the author's investigations lead him to the conclusion that its operations have resulted in a considerable deficit, which is being disguised by the temporary suspension of the sinking fund, by the omission of provision for depreciation, and by "some method by which a part of the cost of operation, maintenance, and general expense, upon the system, is dispersed into the cost of new construction." These conclusions are abundantly supported by figures which include a rather damaging analysis of the financial statement issued by the Ontario Commission in January, 1913.

There appears to be little escape from the author's view that the Commission is selling power not at cost, but at prices considerably below cost as determined upon ordinary business principles, and that the difference is being provided for at the cost of the taxpayers of the Province as a whole. The further extension of loading can only result, in the author's estimation, in a still greater loss; but while this is probably true, we feel that he has scarcely taken proper account of the relief in overhead charges that such increased loading may bring.

Less satisfactory are those portions of the book which deal with the general subject of the development, transmission and utilization of hydro-electric power. The author sees quite clearly that for municipal and similar purposes there is, except in exceptional cases, no economic advantage in hydro-electric power over steam or gas-generated electric power; but when he is found assuming an efficiency of 54 per cent as representative of the transformation of water power into electric power, producing once more the bewhiskered fallacy that the expenditure in travel induced by Niagara Falls is an economic asset, and insinuating, by means the photograph opposite page 64, that a phenomenon which has been observable as a result of low water at Niagara Falls before power developments were undertaken, is "the effect in part, of the expensive experiment on the Canadian Falls," he is writing, not with the knowledge and detached judgment of an engineer, but with the lack of knowledge or prejudice of a public agitator of the dime magazine type.

We deplore these blots on the work all the more since there is no question but that the author has rendered a useful public service in throwing a clear light upon the Ontario Hydro-Electric Commission's position and in dispelling to a considerable degree, by definite figures, the secrecy surrounding its financial operations.